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PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED).

NEW SERIES.

[Ser. 3]

No. 45.

EDITED BY THE SECRETARY.

21494
18. 10. 27

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

31st MARCH, 1922.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

LIST OF OFFICERS, 1922.

President:

George C. Klug, W.A.

Vice-Presidents:

Colin Fraser, S.A. | D. B. Waters, N.Z.

Councillors:

Representing Australasia—
E. G. Banks
G. D. Delprat
Colin Fraser
James Hebbard
James Horsburgh
A. S. Kenyon
John W. Moule
W. E. Wainwright

Representing Queensland—
A. A. Boyd
W. H. Corbould

Representing South Australia—
H. Lipson Hancock

Representing New Zealand—
D. B. Waters

Representing New South Wales—
G H. Blakemore
John M. Bridge
H. A. Evans
C. C. Freeman
F. Danvers Power

Representing Victoria—
Geo. H. Broome
H. Herman
E. W. Skeats

Representing Western Australia—
Richard Hamilton
J. W. Sutherland

Representiug Tasmania—
H. W. Gepp
Robert C. Sticht

Ex officio Member of Council—
R. W. Chapman, S.A. (Past President.)

Hon. Treasurer:

A. S. Kenyon.

Bankers:

Commonwealth Bank of Australia, Collins Street, Melbourne.

Secretary:

D. L. Stirling, Collins Court, 374 Little Collins Street, Melbourne, Vic.

BRANCH SECRETARIES:

Broken Hill -	C. H. J. Clayton, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie -	S. G. Turrell, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan -	B. G. Patterson, Mount Morgan, Q.
Sydney -	F. Danvers Power, 31 Hunter Street, Sydney, N.S.W.

LOCAL CORRESPONDENTS:

South Australia -	H. W. Gartrell, The University, Adelaide.
New Zealand -	D. B. Waters, The University, Dunedin.
Tasmania -	Russell M. Murray, Queenstown.
North Queensland	Murray Russell, Department of Mines, Brisbane.
Papua - -	W. G. Langford, Oilfields, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

Collins Court, 374 Little Collins Street, Melbourne, Victoria.

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Institute Matters.

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MINUTES OF MEETINGS.

OF THE INSTITUTE.

Annual Meeting,

Melbourne, 27th March, 1922,

In the institute rooms, 48 queen street, at 12.30 p.m.

Mr. G. C. Klug (President) occupied the chair.

The Secretary read the notice convening the meeting.

Minutes of the First Ordinary Meeting, August, 1921, and Extra ordinary General Meeting, November, 1921, were confirmed.

The report of the Council and Balance Sheet for 1921 were submitted and adopted.

Messrs. Oxlade and Mackie were elected Auditors for 1922 at a remuneration of £2 2s.

Messrs. E. C. Dyason and C. F. Courtney, on behalf of the members, wished the President, Mr. G. C. Klug, who was shortly leaving for a visit to England, *bon voyage*. Mr. Klug, in returning thanks, said it was his intention, whilst in London, to confer with the Institution of Mining and Metallurgy on matters affecting the welfare of both Institutes.

REPORT OF COUNCIL FOR 1921.

To the Members.

IN presenting the following résumé of the work and progress of the Institute for 1921, members are reminded of the fact that it is the first annual report of the new Council since incorporation was effected.

Membership of the Institute now totals 624, made up as follows :—Life Members, 6 ; Members, 384 ; Associate Members, 169 ; Junior, 1 ; Students, 64. Ten members, 14 associate members, 1 junior, and 7 students were admitted during the year. Eight members and 2 associate members resigned. Seven members, 2 associate members, and 4 students were removed from the register on account of non-payment of subscriptions. Two members died, 1 associate member was transferred to the grade of member. The net gain was 8.

The Council regrets to have to record the loss, by death, of two of its members—Messrs. Henry Battens and Archibald D. Turner. Mr. Battens was admitted to the Institute in 1900, and was, at the time of his death, manager of the Grenville United Mines, Cornwall, England. Mr. Turner was at one time on the staff of the Great Cobar mine, Cobar, N.S.W. He left Australia some years ago for South Africa, where he held a position on the staff of the Falcon Mines Ltd., Umvuma, Rhodesia.

Three meetings of the Institute were held during the year—the Annual Meeting, on 21st March, at Melbourne ; the First Ordinary Meeting, 1921, at Sydney, on 22nd to 27th August ; and an Extraordinary General Meeting at Melbourne on 28th November. The First Ordinary Meeting included excursions to Lithgow, Portland, and Port Kembla, and visits to places of engineering interest in and around Sydney. An account of this meeting was published in Proceedings No. 43. Advantage was taken of the Sydney meeting to further the proposed federation of kindred societies. Progress in regard to this matter is referred to elsewhere in this report.

Vacancies caused by officers retiring in accordance with the rules were filled in November last by the election of the following:—

PRESIDENT: G. C. Klug.

COUNCIL: *Australasia*—Colin Fraser, James Hebbard, A. S. Kenyon, and W. E. Wainwright.
New South Wales—C. C. Freeman and F. Danvers Power.
Victoria—H. Herman.
Queensland—W. H. Corbould.
South Australia—H. Lipson Hancock.
Western Australia—J. W. Sutherland.
Tasmania—H. W. Gepp.
New Zealand—Professor D. B. Waters.

Committees as follow were appointed at the commencement of the year:—An Executive Committee, Finance, Publication, Branches, and Mining Managers' Certificates Committees.

Mr. S. G. Turrell, of 13 Boulder-road, Kalgoorlie, was elected hon. secretary of the Kalgoorlie Branch during the year in place of Mr. F. W. R. Godden, who resigned the position owing to his change of residence. The Council takes this opportunity to record its appreciation of the valuable services rendered to the Institute by Mr. Godden as Branch Secretary for a number of years.

The Broken Hill Branch continued its active and valuable work during the year, under the guidance of Mr. C. C. Freeman (Chairman) and Mr. C. H. J. Clayton (Hon. Secretary), supported by a strong committee. Several interesting papers were read at the meetings of the Branch, and one—viz., "Mechanical Methods of Allaying Dust," by P. H. Warren—was successful in bringing forth a record discussion. Several valuable suggestions regarding mining legislation and other matters of interest to the mining industry were also received from the Branch and endorsed by the Council.

Steps are being taken to establish Branches at Hobart (Tasmania), Port Pirie (S.A.), Mount Lyell (Tasmania), and other centres.

A certificate of incorporation was granted to the Institute on 7th day of April, 1921. A copy of the Memorandum and Articles of Association was published in Proceedings No. 41, and issued to members.

Reference was made in the last annual report to the fact that a movement was on foot to establish some scheme of federation with kindred bodies in Australia. Advantage was taken of the Institute visit to Sydney in August of last year to confer on the subject with the Institution of Engineers and the Australian Chemical Institute. The recommendations framed at that conference were submitted to members for ballot and adopted. They were published in full in Proceedings No. 43. These recommendations, with any modifications which may be suggested by the bodies affected, will be considered at a conference in February, 1922, and their early adoption is anticipated.

A deputation from the Council waited upon the Minister for Mines, N.S.W., and urged a modification of the *Mines Regulations Act* of 1901, which required that all persons starting or stopping pumps and all machinery underground must be certificated. A definite promise was given that the request would be favourably considered. The Minister also agreed to consider in a favourable manner the further suggestion that miners be granted certificates to operate Holman hoists.

An Australian National Council of Research was established under the auspices of the Australasian Association for the Advancement of Science at a meeting of the latter body in January, 1921. The Institute is represented on this Council by Messrs. G. C. Klug and R. C. Sticht.

The establishment of a Federal Geological Survey, which was strongly advocated by the Aus. I.M.M. as far back as 1911, is being pressed by the National Research Council, and a committee was established in August, 1921, to further the movement. The secretary of the committee is Mr. L. Keith Ward, Government Geologist, South Australia, and the Aus. I.M.M. is directly represented on the committee by the President. The committee in its first report emphasizes "the pressing need for the geological survey of hitherto unmapped regions, the systematic determination of the basal geological features of some regions with regard to which some data have been acquired, and the correlation of data in natural regions which transgress the boundaries of individual States."

Difficulty was experienced by mining engineers during the year on account of certain I.M.M. screens not being manufactured true to standard. Three special tests on these screens gave erroneous results. The adoption of such screens in contracts was of little use, and your Council accordingly arranged for the special manufacture and importation of a number of screens true to standard.

The Memorandum of Association of the Institute gives the Council power to establish a register of mining engineers, and, with the establishment of the proposed Federal Engineering Council, it should soon be possible to report progress in this matter. Considerable discussion on the question of registration has taken place during the year in England and America, and strong arguments for and against have been forthcoming. Registration of engineers is already in vogue in Canada and in a number of the States of the U.S.A. In some States registration is optional, in others compulsory. One American society holds "that maintenance of professional standards is regarded as essentially a professional matter; that investors would be more liable to suffer than benefit through licensing; that technical ability is often secondary to other factors, and the matter of qualifications a question of something inherent, or earnest, rather than of something acquired—something that no conceivable licensing system could measure." The Council desires that the question of registration of professional engineers by Act of Parliament, which will shortly come up for review, should not be confused with the proposal to elaborate the register of members of the Aus. I.M.M. by filing a complete list of supplementary qualifications since election.

The publication from time to time of reports in prospectuses by unqualified persons purporting to be mining engineers, geologists, etc., has had a damaging effect on investment in legitimate enterprises, and has cast a reflection on the mining engineering profession. Several glaring cases have been reported. Prospectuses published during last year contained some remarkable statements, and showed either an intentional disregard of facts or a surprising ignorance of the subject dealt with. Steps taken by the Council during the year to remedy this evil have been to a certain degree successful, and it is anticipated that, with the necessary support of the various Governments, the publication of reports by unqualified persons will no longer be permitted.

The Secretary for Mines, N.S.W., has been requested by the Council to convene a conference of Chief Inspectors of Mines for the purpose of drafting a Bill to provide for the standardization of examinations for mining managers' certificates, and reciprocity between the various Australasian States. The Council has endeavoured during the last two years to secure a recommendation from the Premiers' Conferences in regard to this matter, but without success. The more direct course outlined above should expedite this important matter.

The Students' Prize Fund, established by contributions from Messrs. J. W. Sutherland, W. E. Wainwright, and G. H. Blakemore, amounts to £152. The Council decided during the year that from this fund, during 1921, prizes of £10 each be given to student members in each State for the best essays. This fact was made known, but up to the present no essays have been received.

The A. T. Danks prize for the best paper contributed by a student taking part in the visits in connection with the First Ordinary Meeting, 1921, was awarded to G. T. H. M'Mahon, of the Sydney University.

As in 1920, the Imperial College of Science and Technology, South Kensington, London, again offered, in 1921, two Research Fellowships of £300 a year each to aid in carrying out any investigation or research connected with mining, mining geology, metallurgy, or the technology of oil. The Australasian Institute of Mining and Metallurgy acted as the Australian representative of the Imperial College of Science and Technology in this matter.

In April last the Director of the Institute of Science and Industry requested the opinion of the Council on a proposal which he had received for the publication of a book on the mineral resources of Australia. The Council expressed the opinion that such a publication was desirable and necessary, and considered that the work should be undertaken under the control of the Institute of Science and Industry, assisted by a committee representing the various State Mining Departments, the Australasian Institute of Mining and Metallurgy, and similar bodies.

The question of the eligibility of mining accountants of large corporations as members or associate members was discussed, and submitted to members of Council for decision by ballot. The result did not favour their admission.

A bill to provide for the registration of architects was again before the Victorian Parliament during the year, but was abandoned at the close of the session in December. Clauses protecting the rights of mining and other engineers were included, on the recommendation of the Australasian Institute of Mining and Metallurgy and kindred bodies.

A suggestion was submitted by the Broken Hill Branch that the Institute conduct investigations with a view to bringing about improvements in rock-drill steel, to be in keeping with the great advance made in rock-drilling machines. The suggestion was approved, but a scheme for financing the investigation has not yet been evolved.

Admirable arrangements were made by the Broken Hill Branch for the entertainment of students of the Adelaide University during their visit to Broken Hill in August last. A letter of appreciation was received from the University, and the thanks of the Council conveyed to the Branch. The Council is desirous that as much assistance as possible be given to students in this manner.

The Institute was represented at a conference convened by the Commonwealth Advisory Council of Science and Industry, held in Melbourne in December, 1918, to consider the question of engineering standardization. This conference resolved :—(1) That in view of the importance of standardization of engineering materials, it is desirable that such standardization be considered in Australia ; (2) that it is desirable to establish in Australia a representative authoritative body to take the matter in hand ; (3) that it is desirable that the Australian Standards Association be affiliated with the British Association. This matter was revived in November last, when the Council was invited to discuss the subject at a conference to be held in Melbourne in February of 1922. The outcome of this conference will be of importance to the engineering world.

The accounts for the year, which are presented herewith, disclose a sound position. During the year the expenditure was over £400 greater than last year. This was due to the increased amount of publication and the very large amount of non-recurring expenditure —*e.g.*, incorporation expenses, circulars, memoranda, postage, etc., in connection with incorporation and the establishment of a Federal Engineering Council and other charges due to the extension of the activities of the Institute. This necessitated drawing upon the reserve fund to the extent of £138 11s. As the non-recurring expenditure was considerably in excess of this sum, the position is regarded as quite satisfactory.

AUSTRALASIAN INSTITUTE OF MINING AND METALLURGY.

Balance Sheet, 31st December, 1921.

LIABILITIES.

Subscriptions paid in advance				£16	16	0
Donation Trust A/c ...	152	15	0			
Add Interest to 31st December, 1921 ...	7	12	9			
				160	7	9
Life Membership				36	15	0
Sundry Creditors				111	2	4
Accumulated Fund—						
Balance at 31st Dec., 1920	£1956	11	0			
Less Balance transferred from Revenue A/c ...	138	11	0			
				1818	0	0
				£2143	1	1

ASSETS.

State Savings Bank at Credit	£100	0	0			
Commonwealth Bank at Credit	22	2	8			
				£122	2	8
Commonwealth of Australia Inscribed Stock (at cost)	1439	7	6			
Commonwealth of Australia War Savings Certificates (at face value)	281	0	0			
				1720	7	6
Outstanding Subscriptions				232	0	0
Suspense Account				25	0	0
Office Furniture, Melbourne and Broken Hill Branch				43	10	11
				£2143	1	1

Melbourne, 3rd *March*, 1922.

Audited and found correct.

Oxlade & Mackie, Auditors.

Statement of Receipts and Expenditure for the Year ended 31st December, 1921.

RECEIPTS.

Dec. 31st, 1920.						
To Commonwealth Savings Bank at Credit ...	£66	19	1			
State Savings Bank at Credit	234	9	7			
				£301	8	8
Dec. 31st, 1921.						
Members' Subs., 1919 ...	4	4	0			
,, ,, 1920 ...	52	17	11			
,, ,, 1921 ...	723	17	0			
,, ,, 1922 ...	11	11	0			
,, ,, 1923 ...	1	1	0			
				793	10	11
Life Membership ...	21	0	0			
Entrance Fees	45	15	0			
Advertisements ...	54	0	0			
Sales	9	16	0			
Interest--Commonwealth Bank	4	2	3			
Interest—State Savings Bank	9	7	2			
Interest—Inscribed Stock	96	0	0			
				240	0	5
				£1335	0	0

EXPENDITURE.

Dec., 31st, 1921.						
By Printing & Stationery ...	£85	15	6			
Rent	50	0	0			
Salaries	276	0	0			
Postages, Telegrams, &c.	99	5	8			
Sundries	49	15	2			
Proceedings	381	6	4			
Commission	5	18	6			
Meeting Expenses ...	86	6	2			
Suspense A/c	25	0	0			
Branch Expenses ...	25	0	0			
Incorporation Expenses	32	10	0			
				1116	17	4
Commonwealth of Australia War Savings Certificates (at face value)	...		...	96	0	0
Commonwealth Bank at Credit	22	2	8			
State Savings Bank at Credit	100	0	0			
				122	2	8
				£1335	0	0

REVENUE ACCOUNT FOR THE YEAR ENDED 31ST DECEMBER, 1921.

1921.				1921.			
Dec. 31.	To Publications	£381 6 4		Dec. 31.	By Subs., 1919 ...	£4 4 0	
	Stationery & Printing ...	194 10 10			,, 1920 ...	93 0 0	
	Postages, Telegrams, &c....	99 5 8			,, 1921 ...	765 17 0	
	Salaries ...	276 0 0			Advertisements	44 0 0	
	Rent ...	50 0 0			Sales ...	9 16 0	
	Sundries ...	52 2 2			Entrance Fees	45 15 0	
	Commission	5 18 6			Interest ...	109 9 5	1072 1 5
	Meeting Expenses ...	86 6 2			Balance transferred to Accumulated Fund ...		138 11 0
	Branch Expenses ...	25 0 0					
	Incorporation Expenses...	32 10 0					
	Interest on Donation Trust A/c...	7 12 9					
			£1210 12 5				
			£1210 12 5				£1210 12 5

Melbourne, 3rd March, 1922. Audited and found correct. OXLADE & MACKIE, *Auditors.*

OF THE COUNCIL.

(Summary.)

JANUARY 16TH, 1922—1 P.M.

Resolved that Professor D. B. Waters be elected Vice-President for one year and Colin Fraser for two years.

Resolved that Mr. A. S. Kenyon be elected Hon. Treasurer.

Resolved than an Executive Committee be appointed, as provided by Article 61.

Other committees, as follow, were appointed for the year:—*Publication and Education*: Messrs. Colin Fraser, H. W. Gepp, H. Herman, G. H. Broome, W. E. Wainwright, J. M. Bridge, and Professor Skeats; *Finance*: Messrs. A. S. Kenyon, H. Herman, and Colin Fraser; *Branches*: Messrs. R. C. Sticht, G. H. Broome, J. M. Bridge, W. E. Wainwright, and the Chairman of the Branch; *Mine Managers' Certificates*: Messrs. A. H. Merrin, H. Herman, and G. H. Broome.

The Secretary's report was presented, and showed that receipts since last meeting were:—Subscriptions, £85; interest, £40; advertisements, £12; and sales, £1 8s.

Accounts to the amount of £121, including printing and process blocks, £108, and petty cash advance, £10, were passed for payment.

Nominations were submitted and dealt with as follows:—As Members—Samuel Leah, disapproved; Henry Lovett Cameron, approved; John Allen Cameron, deferred. Associate Members—Percy H. O'Nians, approved; Arthur Delmar Combe, approved; H. H. Robbins, deferred. Mr. I. B. Fabrikant was elected an Associate Member.

Resignations of Messrs. E. A. Weinberg and A. Mica Smith were accepted.

Papers as follow were referred to the Publication Committee:—"Gold Occurrences at Bendigo," by Dr. F. L. Stillwell; "Electrical Power Application on the Broken Hill Mines," by F. J. Mars.

It was reported that the A. T. Danks Prize, 1921, had been awarded to G. T. H. M'Mahon, of the Sydney University.

Messrs. G. C. Klug, A. S. Kenyon, W. E. Wainwright, John M. Bridge, G. H. Broome, and H. Herman were appointed delegates to the conference on engineering standardization to be held in Melbourne on 3rd February.

Messrs. G. C. Klug and A. S. Kenyon were appointed to represent the Institute at the conference *re* proposed Federal Council to be held on 2nd February.

It was reported that the Architects' Registration Bill (Vic.) 1921 was abandoned at the close of the 1921 session.

Mr. S. G. Turrell, of Kalgoorlie, was elected hon. secretary of the Kalgoorlie Branch.

Resolved that a meeting of the Full Council be held on the same day as the Annual Meeting—viz., 27th March, 1922.

Consideration of the matter of the First Ordinary Meeting, 1922, was deferred.

Resolved that the action of the Council in the matter of prospectuses be reported to the Federal Council when constituted.

Suggestions from the Broken Hill Branch in regard to Register of Members were referred to a Special Committee.

OF THE EXECUTIVE COMMITTEE.

FEBRUARY 6TH, 1922.—1 P.M. (SPECIAL MEETING).

A report of the conference *re* proposed Federal Council, held in Melbourne on 2nd February, was submitted. It was resolved that the recommendations be accepted; that, as the alterations to the original recommendations (contained in circular No. 5, 1921) are of an unimportant character, it is not necessary to re-submit the matter to ballot of members; that, as provided, three representatives of the Council of the Aust. I.M.M. be appointed at a meeting of the Council to be held on 6th March.

A report of the conference *re* Engineering Standardization, held in Melbourne on 3rd February, was submitted. It was resolved that the report be received and adopted, and that the appointment of Mr. W. E. Wainwright as a member of the Advisory Committee be approved.

Action taken by the President and Secretary to prevent the removal of the Geological Survey Museum (Vic.) from its present location alongside the Mines Department was endorsed.

MARCH 6TH, 1922.—1 P.M.

The Secretary's report was presented, and accounts to the amount of £85, including salary and rent for two months, £50, lithos., £16, process blocks, £3, and petty cash, were passed for payment.

Communications *re* Federal Engineering Council from members of Council showed that they were generally in favour of the recommendations made to date.

Official copies of the report of the conference on 3rd February *re* Engineering Standardization were submitted. It was explained that advance copies had already been issued to members of Council of the Australasian Institute of Mining and Metallurgy.

A report of the annual meeting of the Broken Hill Branch was submitted, and election of officers confirmed.

Applications for admission to the Institute were submitted and approved:—As Members—Alex. George Campbell (transfer from Associate), Richard Francis Hockey, Henry Alex. Vaudeau. As Associate Members—Eben Tracy Francis, Frederick Albert Roberg. As Junior—Robert Thomas Goodyear. As Student—Robert Pittman Hooper (admitted).

Resignations of Messrs. W. J. Rose, R. S. Black, and N. S. K. M'Callum, on account of their having left the mining industry, were accepted.

Deaths of Messrs. J. W. Ashcroft and C. T. Townsend were reported.

A draft Bill providing for Registration of Engineers was submitted from the Institution of Engineers. Resolved that copies be supplied to members of Council, with a request for opinions and suggestions.

Resolved—

(1) That the Institute ask the Government to take the necessary steps to render it obligatory that persons making technical reports for the promotion of public companies present their credentials to the Government before so doing. Such credentials to clearly and fully state what they represent.

(2) That the Institute urge the immediate necessity for a mining companies law, especially relating to mining prospectuses, and providing that directors shall be personally liable for any misleading statements contained in such prospectuses.

Resolved that a meeting to revise the Annual Report of the Council be held a fortnight hence.

Mr. W. E. Wainwright was nominated to the Engineering Faculty, Melbourne University, in place of Mr. G. C. Klug, who is shortly leavng for a visit to England.

MARCH 20TH, 1922.—1 P.M. (SPECIAL MEETING).

Draft of Annual Report of Council for 1921 and copy of audited Balance Sheet were submitted, and, with a few amendments to the Report, were approved.

The following were appointed delegates to represent the Institute on the Federal Engineering Council:—Mr. G. C. Klug (in the event of Mr. Klug being unable to act, Mr. W. E. Wainwright to be appointed), Mr. F. Danvers Power, and Mr. G. H. Blakemore.

A draft Bill providing for Registration of Professional Engineers, received from the Institution of Engineers, Australia, was referred to. It was resolved that the I.E.A. be informed that this important matter is under consideration by the Council of the Aus. I.M.M., who would be glad to know if it is the intention of the Council of the I.E.A. to submit the matter to the Federal Council when the present draft has been amended to the satisfaction of members of the I.E.A.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays an Public Holiays.

MEMBERSHIP.

Since the issue of Proceedings No. 44 the following have been admitted to the Institute :—

Members.

Names and Addresses.

CAMERON, HENRY LOVETT, 5 Hamilton-street, Sydney, N.S.W.
CLINTON, ROBERT ADAM, Curry-street, Merewether, N.S.W.
HOCKEY, FRANCIS HENRY, Broken Hill Proprietary Co., Whyalla, S.A.
VAUDEAU, HENRY ALEXANDER, Zeehan, Tas.

Transfer—Associate Member to Member.

CAMPBELL, ALEXANDER GEORGE, Broken Hill Pty. Block 10 Co., Broken Hill, N.S.W.

Associate Members.

COMBE, ARTHUR DELMAR, Geological Survey, Entebbe, Uganda.
FABRIKANT, ISIDORE B., 50 Erskine-street, North Melbourne, Vic.
FRANCIS, EBEN TRACY, Amalgamated Zinc (De Bavay's) Ltd., Broken Hill, N.S.W.
O'NIANS, PERCY HENRY, c/o Anglo-South American Bank Ltd., Mexico City, Mexico.
ROBERG, FREDERICK ALBERT, Broken Hill Pty. Block 10 Ltd., Broken Hill, N.S.W.

Junior.

GOODYEAR, ROBERT THOMAS, 181 Chloride-street, Broken Hill, N.S.W.

Students.

GROSS, KLEM BASSETT, 21 Esmond-street, Unley, S.A.
HOOPER, ROBERT PITMAN, Broken Hill South Ltd., Broken Hill, N.S.W.
ELFORD, HAROLD STEWART, 19 Beulah-road Extension, Norwood, S.A.

NEW ROOMS.

Members are reminded that the Head Office of the Institute is now at Collins Court, 374 Little Collins-street, Melbourne. The new quarters are modern, roomy, and well lighted, and are located in a suitable part of the city. Under the same roof are the Australian Mines and Metals Association, the Australian Metal Exchange, the Gold Producers' Association, and the Chamber of Mines of Victoria.

THE FEDERAL ENGINEERING COUNCIL, AUSTRALIA.

A further conference between representatives of the Australasian Institute of Mining and Metallurgy (Inc.), the Institution of Engineers, Australia, and the Australian Chemical Institute was held in Melbourne on 2nd February. At this conference a few unimportant amendments were made to the recommendations as submitted to and accepted by members of the Aus. I.M.M. The recommendations have been formally accepted by the Council. The Institute's delegates on the Federal Council are Messrs. G. H. Blakemore, F. Danvers Power, and W. E. Wainwright.

ENGINEERING STANDARDIZATION.

The Institute was represented at a conference held in Melbourne on 3rd February to consider the formation of an Engineering Standards Association. Several important recommendations were made. These recommendations will shortly be submitted to the Minister of Trade and Customs and then published.

REGISTRATION OF PROFESSIONAL ENGINEERS.

A draft Bill providing for the registration of professional engineers has been received from the Institution of Engineers, Australia, and is now under consideration by the Council. As the mining engineer is particularly interested in any such legislation, the Council of the Aus. I.M.M. will make every effort to safeguard the interests of members.

REPORTS IN PROSPECTUSES.

At a recent meeting of the Executive Committee the following resolutions were carried :—

(1) That the Institute ask the Government to take the necessary steps to render it obligatory that persons making

technical reports for the promotion of public companies present their credentials to the Government before so doing; such credentials to clearly and fully state what they represent.

(2) That the Institute urge the immediate necessity for a mining companies law, especially relating to mining prospectuses, and providing that directors shall be personally liable for any misleading statements contained in such prospectuses.

MOVEMENTS OF MEMBERS.

Percy H. Baldwin has left Mount Morgan. His address now is Chelmsford-avenue, Wooloowin, Brisbane, Queensland.

W. E. Barron has commenced private practice as a consulting mining and metallurgical engineer, his address being c/o National Bank of South Africa, East London, South Africa. Early in the year he was in the South-West Africa Protectorate investigating a tin proposition.

J. D. Connor has been appointed Principal of the Wonthaggi Technical School, Wonthaggi, Vic.

H. James Daggar sailed for England *via* America on 8th February.

E. B. Dow is severing his connection with the Sulphide Corporation Ltd., Cockle Creek, after 2½ years' service. He is taking the position of surveyor to the Block 10 (Misima) G.M. Co. N.L., Misima, Papua.

A. L. Hay is leaving for Papua on 1st April. His address will be c/o Block 10 (Misima) G.M. Co., Misima, Papua.

Mr. **G. C. Klug**, President Aus. I.M.M., is leaving Melbourne in April for a visit to London. He expects to be absent from Australia for about six months.

Percy H. O'Nians, of Waihi, New Zealand, has sailed for Mexico. His address now is c/o Anglo-South American Bank Ltd., Mexico City, Mexico.

J. R. Peberdy has accepted an appointment on the staff of the National Portland Cement Co. at Maria Island, Tasmania.

L. A. Westcott, who has for a number of years been on the staff of the Mount Morgan Gold Mining Co. Ltd., has been appointed assistant blast furnace manager at the Lithgow iron and steel works.

W. E. Wainwright (member of Council) has been appointed to represent the Aus. I.M.M. on the Engineering Faculty, Melbourne University.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY
FROM 31ST DECEMBER, 1921 TO 31ST MARCH, 1922.

Australian Mining and Engineering Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Chamber of Mines of Western Australia : Journal	quarterly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
Victorian Department of Mines : Records of the Geological Survey, Vol. IV., Part 3		Melbourne
South Australia Department of Mines : Review, Nos. 33, 34		Adelaide
New Zealand Board of Science and Art : Journal, Vol. IV., No. 5		Wellington
New Zealand Geological Survey : Bulletin, No. 23		Wellington

Institution of Mining Engineers:	
Transactions, Vol. LXI., Part 6; LXII., Part 3	London
Institution of Mining and Metallurgy:	
Bulletin, No. 208	London
Geological Society:	
Quarterly Journal, No. 308	London
Geologists' Association:	
Proceedings, Vol. XXXIII., Part 1	London
Imperial Mineral Resources Bureau:	
Reports on Mica	London
Société des Ingeneiurs Civils de France:	
Bulletins, April to June, 1921	Paris
Department of Mines, Canada:	
Geological Survey Report, 1920, D.	Ottawa
Department of the Interior:	
Tenth Annual Report of the Director Bureau of Mines	
Bulletin, No. 184—Manufacture of Sulphuric Acid in the United States	
Technical Paper, No. 266—Coke Oven Accidents	Washington

RECENT ARTICLES ON MINING MATTERS.

(31st December, 1921, to 31st March, 1922).

Note.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Mining and Engineering World*, Chicago, wk., 10c.

(6) *The Engineering and Mining Journal*, New York, wk., 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining and Scientific Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey of India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*, Inst. Min. Eng., London.

(27) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. & Met. Eng. New York City.

(30) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Australian Mining and Engineering Review*, Melbourne, Vic., mth., 6d.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

(35) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London.

(38) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

LIST OF ARTICLES.

GEOLOGICAL.

The Origin of Primary Ore Deposits. P. W. Whitehead and D. K. F. MacLachlan. (35) Jan., 1921.
A Typical Example of Magmatic Injection. J. F. Allan. (25) Bull. No. 208.

MECHANICAL.

Mechanical Mining of Anthracite. H. D. Kynor. (29) Sept., 1921.
Power Installation at Coverdale Mine. C. M. Means. (29) Sept., 1921.
Determination of Electrical Equipment for a Mine Hoist. G. Bright. (29) Sept., 1921.
Underground Loading Devices in Metal Mines. C. L. Colburn. (8) Jan. 28, 1922.
Scraping and Loading in Mines with Small Compressed-Air Hoists. W. Royce. Serial commenced Dec. 10, 1921.
Recent Advance in Design of Gold-Dredge Bucket Lips. G. J. Young. (6) Dec. 10, 1921.
Belt Gear and Chain Drives for Grinding Mills. G. J. Young. (6) Dec. 17, 1921.
A Method for Measuring Leakage in Compressed Air Lines. W. S. Weeks. (6) Jan. 7, 1922.
Uses of Compressed Air in Mining. W. R. Wade. (6) Jan. 28, 1922.
The Feld Scrubber for Cleaning Metallurgical Smoke. W. F. Lamoreaux. (6) Feb. 4, 1922.
Tension in Winding Ropes. J. Stoney. (26) Vol. LXII., Part 1.
Electric Welding of a Cast-Iron Rope-Drive Flywheel. N. E. Webster. (26) Vol. LXII., Part 1.
Notes on an Application of the Cementation Process to a Battery Foundation. B. Schlesinger. (28) Nov., 1921.
Application of Pulverized Coal to Boilers. J. W. Fuller. (29) Dec., 1921.
Storage Battery Locomotive as Applied to Mine Haulage. C. E. Stuart. (29) Jan., 1922.
Advances in the Preparation of Anthracite. D. C. Ashmead. (29) Sept., 1921.
Heat Treatment of Rock-Drill Steel. (29) Sept., 1921.

METALLURGICAL.

Flotation of Pyrite. W. S. Morley. (29) Aug., 1921.
Development of the Disc-Crusher and Leaching Practice. A. W. Allen. (8) Dec. 17, 1921.
The Lead-Acetate Tartaric-Acid Method of Assaying Cyanide Solutions. F. Borzynski. (8) Dec., 1921.
Ball-Milling and Flotation at Gatema, Chile. F. Benitez. (8) Serial commenced Dec. 24, 1921.
Historical Sketch of Ore Treatment in Colorado. H. F. Lunt. (8) Dec. 24, 1921.
Surface-Tension in Flotation. A. C. Halferdahl. (8) Jan. 21, 1922.
Reclaiming Six-Ounce Silver Tailing at the Coniagas Mines. H. E. Cawley. (6) Dec. 3, 1921.
Notes on Lead Bullion Refining. G. G. Griswold. (6) Dec. 17, 1921.
Killing Flotation Froth. F. A. Sundt. (6) Dec. 17, 1921.
The Santa Barbara 500-Ton Mill for Concentrating Lead Carbonate Ores. F. M. Heidelberg. (6) Dec. 31, 1921.
Flotation Tests on Zinc Table Slime. M. H. Thornberry. (6) Jan. 14, 1922.
Calculation of Anode-Furnace Charges in a Copper Refinery. C. M. Brister. (6) Jan. 28, 1922.
The Friction Co-efficient of Minerals. S. Nettleton. (26) Vol. LXII., Part 1.
Automatic Cone Classifier Underflow Regulator. H. Brazier. (28) Nov., 1921.
The Correlation of Metallurgical Statistics. H. A. White. (28) Dec., 1921.

MISCELLANEOUS.

Mine Fires Extinguished by Sealing. D. Bunting. (29) Aug., 1921.
Anthracoal: a New Domestic and Metallurgical Fuel. D. Markle. (29) Aug., 1921.
Octagonal Ventilation Shaft of Davis–Daly Copper Company. J. L. Bruce. (29) Sept., 1921.
Relation of Gypsum Supplies to Mining. D. H. Newland. (29) Sept., 1921.
Lynch Plant of United States Coal and Coke Company. H. N. Edvenson. (29) Sept., 1921.
Ventilation and Working Efficiency. B. W. Holman. (35) Serial commenced Nov., 1921.
Relation of Mining to the Development of the State. L. D. Ricketts. (8) Jan. 7, 1922.

Magnetic Prospecting. W. S. Weeks. (8) Jan. 21, 1922.
Prospecting for Oil with the Diamond Drill. J. S. Mitchell. (6) Jan. 7, 1922.
The New Stack at the Midvale Smelter. D. W. Jessup. (6) Jan. 14, 1922.
Licensing the Mining Engineer. (6) Jan. 28, 1922.
A Method of Measuring Ventilating Resistances. D. Penman. (26) Vol. LXII., Part 1.
The Testing of Anemometers. J. Cooper. (26) Vol. LXII., Part 1.
The Operation of Fans in Parallel. J. Parker. (26) Vol. LXII., Part 3.
Diamond Drill Sampling Methods. R. D. Longyear. (29) Dec., 1921.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED)

NEW SERIES.

No. 46.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

30th JUNE, 1922.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

LIST OF OFFICERS, 1922.

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Vice-Presidents:

Colin Fraser, S.A. | D. B. Waters, N.Z.

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BRANCH SECRETARIES:

Broken Hill - C. H. J. Clayton, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie - S. G. Turrell, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan - B. G. Patterson, Mount Morgan, Q.
Sydney - F. Danvers Power, 31 Hunter Street, Sydney, N.S.W.

LOCAL CORRESPONDENTS:

South Australia - H. W. Gartrell, The University, Adelaide.
New Zealand - D. B. Waters, The University, Dunedin.
Tasmania - Russell M. Murray, Queenstown.
North Queensland - Murray Russell, Department of Mines, Brisbane.
Papua - W. G. Langford, Oilfields, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

Collins Court, 374 Little Collins Street, Melbourne, Victoria.

Institute Matters.

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MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Summary.)

MAY 15TH, 1922.—1 P.M.

The death of Mr. R. C. Sticht, Member of Council and Past President, was reported. Great regret was expressed at the loss the profession had sustained. The Secretary was directed to include an obituary notice in the next issue of Proceedings, and to forward an expression of sympathy to Mrs. Sticht on behalf of the Council and members.

Minutes of the previous meetings (2) were read and confirmed.

The Secretary's report was presented, and showed that receipts since 1st January, 1922, were £481.

Accounts to the amount of £81 were passed for payment.

Nomination of Mr. Henry James Truscott as an Associate Member was approved for ballot. Robert Belshaw Paterson was admitted as a Student.

Resignations of Messrs. C. L. Irvine and C. N. Ross on account of their having left the mining industry were accepted.

A paper by H. H. Carroll entitled "Ventilation and the Kata Thermometer" was referred to the Publication Committee.

Resolved that the Institution of Engineers, Australia, he informed that the Aus. I.M.M. approves of the suggestion of the Executive Committee of the Council of the I.E.A., viz., that the proposed draft Bill providing for Registration of Engineers be referred to the Federal Engineering Council for consideration and report. It was further resolved that suggestions on the Bill received from members of Council of the Aus. I.M.M. be collated and forwarded to all members of Council.

Engineering Standardization.—A draft memorandum on this subject for presentation to the Minister of Trade and Customs was submitted and approved. Resolved that the Aus. I.M.M. hold itself responsible for £100 for a fund to initiate the scheme, this amount to be obtained, if possible, from the Australian Mines and Metals Association, to whose members (Mining Companies) the proposal was of special importance financially.

JUNE 26TH, 1922.—1 P.M.

Accounts to the amount of £131, including Printing Proceedings £94, Photo Blocks, Transfers, etc. £22, Postage and Sundries £20, were passed for payment.

Nomination of (1) Mr. Edward Finch (as Member) was deferred, (2) Ernest F. Harris (as Member) was held over, and of Henry James Truscott (as Associate Member) was referred to a special sub-committee to report.

John Arthur Blackett Forster was admitted as a Student.

Resolved that copies of suggestions sent by the Institute to Departments of Mines *re* reports in Prospectuses and liability of Directors for statements in Prospectuses be forwarded to members of the Executive Council, and that a special meeting be held in a fortnight's time to consider the matter.

It was reported that the Department of Mines, N.S.W., had decided to defer, for the present, consideration of the Institute's suggestion to convene a conference of Chief Inspectors of mines to deal with Standardization of Mining Managers' Certificates.

A report of a meeting of the Broken Hill Branch was submitted, and arising therefrom it was resolved that the Council endorse the Branch's suggestion to appeal for a geological survey of north-west New South Wales.

The Secretary was instructed to explain to the Broken Hill Branch the position in regard to certain papers on local subjects published in the Proceedings before being discussed by Branch members.

Further discussion took place on the question of Registration of Professional Engineers. Resolved that, pending receipt of opinions desired from other members of Council, further consideration be deferred.

Engineering Standardization.—Mr. Wainwright reported deputation to Minister of Trade and Customs on 26th May, when a memorandum on this subject was submitted and favorable consideration promised.

A donation by Mr. C. F. Courtney of bound copies of the Journal of the Royal Society of Arts and of Minutes of Proceedings of the Institute of Civil Engineers was accepted with thanks.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holiays.

FIRST ORDINARY MEETING, 1922.

Arrangements are being made for an Institute visit to Port Pirie towards the end of September, 1922, when the first ordinary meeting will be held. Members are advised to make their other appointments fit in with this important visit. Concessions in travelling fares will be arranged for those notifying their intention to be present. A programme giving details will be issued later.

PUBLICATIONS.

Papers appearing in this issue are :—

- Electrical Power Application on the Broken Hill Mines, with particular Reference to Remote Motor Control and Electrical Hoisting at the South Mine. By F. J. Mars.
- * Burma Corporation Ltd. A Description and Review of Early Chinese and Present Day Operations. By John W. Moule.
- Ventilation and the Kata Thermometer. By H. H. Carroll (with discussion).

Discussion on each of the above papers or on any other papers appearing in the Proceedings will be welcomed, and where Branches exist evenings might be reserved for discussion on such Papers.

ENGINEERING STANDARDIZATION.

A memorandum on the subject of Engineering Standardization has been prepared by a special sub-committee, on which the Aus.I.M.M. is represented, and was submitted to the Minister of Trade and Customs on 26th May. The Minister promised favorable consideration of the matter.

REGISTRATION OF PROFESSIONAL ENGINEERS.

This subject is under discussion by the Council, and the possible effect of registration as proposed is being carefully considered in the interests of members.

REPORTS ON PROSPECTUSES.

A special meeting of the Executive Committee will shortly be held to consider the question of introducing legislation to control reports on prospectuses by unqualified men and to define the liability of Directors for misleading statements in such prospectuses.

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* This paper was received in January, 1922. Publication was postponed pending receipt of permission from the Indian Geol. Survey to include certain illustrations.

DONATION.

Members are indebted to Mr. C. F. Courtney for a valuable donation of copies of Minutes of Proceedings of the Institution of Civil Engineers and of the Journal of the Royal Society of Arts.

MOVEMENTS OF MEMBERS.

BOYD BENNIE, for some year's Inspector of Mines, N.Z., has retired from Government service and is now practising as a mining engineer. His address is—First Avenue, Whangaru, N.Z.

G. A. DENNY has returned to South Africa. His present address is Box 13, P.O., Bulawayo, Rhodesia.

CLEMENT DIXON's present address is—c/o Cyprus Asbestos Co. Ltd., Mine Office, Amiandos on Troodos, via Limassol, Cyprus.

I. B. FABRIKANT has been appointed manager of the London Christmas Gold Mine at Bendigo.

GEO. B. GAME has joined the staff of the Victorian State Rivers and Water Supply Commission and is stationed at the Eildon Weir, via Alexandra, Vic.

GEO. J. GRAY, late of Laloki Mine, Papua, is now in Melbourne.

JOSEPH GRIGG has returned to the Federated Malay States and is with the Malayan Prospecting Syndicate Ltd., Kuantan, via Singapore.

ERNEST K. HALL has been appointed manager of the Christmas Gift G. M. Co. Nth., Cullinga Mine, via Wallendbein, N.S.W.

W. KINGSTON has resigned from the position of Director of the Bairnsdale School of Mines, which he has held for a number of years, and is now General Manager of the Mount Quamby G.M.Co., Cloncurry, Queensland.

KEITH LEWIS left Australia to join the staff of the Raub Australian Gold Mines at Pahang, F.M.S.

GEORGE A. MORE, of Singapore, is at present visiting Australia.

F. M. MURDOCH is now with the Waterloo Chemical Works Ltd., Hobart, Tas.

RUSSELL M. MURRAY is now General Manager of the Mount Lyell Mining and Railway Co. Ltd.

W. J. NICOL has been appointed Manager of the United Gleeson's G.M. Co. at Gaffney's Creek, Vic.

H. J. ROBERTSON, late of the Marion Bay Gypsum Works has left Australia to take charge of operations at the Bibiani Gold Mines, Gold Coast, West Africa.

BRANCH REPORTS.

BROKEN HILL BRANCH.

The Annual General Meeting of the Broken Hill Branch was held at the Protestant Hall, Broken Hill, on Tuesday evening, 28th February, 1922.

The Annual Report and Balance Sheet were submitted and adopted, and the Institute's activities reviewed by the Hon. Secretary.

The following were elected office-bearers for the ensuing year :—

Chairman—C. C. Freeman.
Vice-Chairmen—W. D. Whitemaan & F. Voss Smith.
Hon. Secretary—C. H. J. Clayton.
Hon. Treasurer—P. H. Warren.
Hon. Auditor—H. W. S. Cherry.
Committee—A. Clark, H. J. Harrison, R. D. Nevitt, W. D. Whiteman, E. T. Henderson, V. A. Haigh, N. J. Roberts, F. N. Thomas and W. B. Chomley.

A Paper entitled "Ventilation and the Kata Thermometer," by H. H. Carroll. was read and discussed.

Resolved that Mr. C. H. J. Clayton's paper on "The Contract System in Broken Hill" be presented at the next meeting.

An Ordinary General Meeting of the Broken Hill Branch was held on Tuesday, June 20th, 1922, at 8 p.m.

Mr. C. C. Freeman presided.

Resolved that the General Secretary be asked why papers of local interest had been published in the Proceedings before being discussed at the Branch meeting.

The Chairman reminded members of the Institute visit to Port Pirie in September next.

Part I. of a paper by P. H. Warren and T. A. Read describing the methods and apparatus for determination of dust in air was read by Mr. Warren. The following exhibits were handed round:—The Modified Kotze Ejector, Read Water Spray Dust Collector, Brass Sugar Tubes, old and new style Konimeters attached to Microscopes and complete working drawings of all apparatus. These exhibits added greatly to the interest in the subject of the paper.

It was resolved that the Branch convey to Mr. E. C. Andrews, B.A., its congratulations on the publication of his report on the Geology of the Broken Hill District, also that an appeal be made for the extension of the survey to cover the whole of the north-western district of New South Wales. It was further resolved that the Council be requested to support the appeal.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 31st March, 1922 to 30th June, 1922.

Australian Mining and Engineering Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal	weekly	New York
Iron and Coal Trades Review	weekly	London
Mining and Engineering World	weekly	Chicago
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Chamber of Mines of Western Australia : Journal	quarterly	Kalgoorlie
The West Australian Mining, Building and Engineering Journal	weekly	Perth
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
Victorian Department of Mines : Bulletin Geological Survey, No. 45		Melbourne
Royal Society of Victoria : Proceedings, Vol. XXXIV., Part II.		Melbourne
Department of Labour and Industry, N.S.W. : Reports of Technical Commission on Industrial Diseases at Broken Hill		Sydney

Royal Society of Queensland :	
Proceedings, Vol. XXXIII.	Brisbane
Tasmanian Department of Mines :	
Underground Water Supply, Paper No. 2	Hobart
Royal Society of Tasmania :	
Papers and Proceedings, 1921	Hobart
New Zealand Board of Science and Art :	
Journal, Vol. IV., No. 6; Vol. V., Nos. 1 and 2	Wellington
Institution of Mining Engineers :	
Transactions, Vol. LXII., Part 4; Vol. LXIII., Parts 1 and 2	London
Institution of Mining and Metallurgy :	
Bulletins, Nos. 209-212	London
Geological Society :	
Quarterly Journal, No. 309	London
Geologists' Association :	
Proceedings, Vol. XXXIII., Part 2	London
Iron and Steel Institute :	
Journal, Vol. CIV., Part 2	London
Department of Scientific and Industrial Research :	
First Report of the Mine Rescue Apparatus Research Committee; Report of the Fuel Research Board for 1918 and 1919; Memorandum on Solid Lubricants; Electro-Deposition of Iron	London
Imperial Mineral Resources Bureau :	
Reports on Gold, Tin, Lead, Sulphur and Iron Pyrites, Platinum and Allied Minerals, Copper Third Annual Report Mining Laws, Vol. IV., Canada	London
Royal Dublin Society :	
Scientific Proceedings, Vol. XVI., Nos. 14-34	Dublin
Societa Geologica Italiana :	
Bulletin, XL., Part 3	Rome
Société des Ingeneiurs Civils de France :	
Bulletins, July to Dec., 1921	Paris
Annales des Mines :	
Vol. XII., Parts 10-12, 1922; Vol., I., Parts 1-4, 1922	Paris
Mining and Geological Institute of India :	
Transactions, Vol. XVI., Part 2	Calcutta

Geological Survey of India :	
Records, Vol. LIII., Part 2 - - -	Calcutta
Department of Mines, Canada :	
Annual Report, 1920 - - -	Ottawa
Nova Scotian Institute of Science :	
Proceedings and Transactions, Vol. XV., Part 1 -	Halifax
Bureau of Sciences, Manila :	
Mineral Resources of the Phillipine Islands, 1919 and 1920 - - - - -	Manila
Secretaria de Industria Y Comercio :	
Boletin Dept. of Mines, Vol. XII., Nos. 4-6,; Vol. XIII., No. 1 - - - -	Mexico
Association Peruana :	
Archivos, Vol. I, Part 1 : - -	Lima

RECENT ARTICLES ON MINING MATTERS.

(31st March, 1922 to 30th June, 1922.)

Note.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Mining and Engineering World*, Chicago, wk., 10c.

(6) *The Engineering and Mining Journal*, New York, wk., 15c.

(7) *The Colliery Engineer*, Scranton, Pa., U.S.A., mth., 20c.

(8) *Mining and Scientific Press*, San Francisco, Cal., wk., 10c.

(9) *Annales des Mines*, Paris, France, mth.

(10) *Publications*, Department of Mines, Melbourne, Victoria.

(11) *Publications*, Department of Mines, Sydney, New South Wales.

(12) *Publications*, Department of Mines, Adelaide, South Australia.

(13) *Publications*, Department of Mines, Brisbane, Queensland.

(14) *Publications*, Department of Mines, Perth, Western Australia.

(15) *Publications*, Department of Mines, Hobart, Tasmania.

(16) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(17) *Publications*, Bureau of Mines, Toronto, Ontario.

(18) *Publications*, Geological Survey of India, Calcutta.

(19) *Publications*, Geological Survey, U.S.A., Washington.

(20) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(21) *Publications*, California State Mining Bureau, Sacramento, Cal.

(22) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(23) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(24) *Quarterly Journal*, Geological Society, London.

(25) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.

(26) *Transactions*, Inst. Min. Eng., London.

(27) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(28) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(29) *Transactions*, Am. Inst. of Min. & Met. Eng. New York City.

(30) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(31) *Journal*, Franklin Inst., Philadelphia, Pa.

(32) *Australian Mining and Engineering Review*, Melbourne, Vic., mth., 6d.

(33) *Transactions*, Am. Soc. C.E., New York City.

(34) *Bulletins*, Société des Ingénieurs Civils, Paris.

(35) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(36) *Publications*, Iron and Steel Institute, London.

(37) *Proceedings*, Inst. of Mech. Eng., London

(38) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(39) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(40) *Transactions*, Mining and Geological Institute of India, Calcutta.

(41) *Publications*, Department of Mines, Wellington, N.Z

(42) *Journal*, Chamber of Mines of West Australia, Perth.

(43) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

LIST OF ARTICLES.

GEOLOGICAL.

Ore Deposits of the Pioche District, Nevada. J. C. Anderson. (6) Feb. 18, 1922.
Platinum in Quartz-Veins. H. W. Turner. (6) March 25, 1922.
Geology and Mining of Mica. J. V. Lewis. (6) May 20, 1922.
Copper in the Sierra Nevada, Spain. C. De Kall. (8) March 4, 1922.
Work of the Geological Department at a Porphyry Copper Mine. X. B. Starnes. (8) Serial commenced March 18, 1922.
The Geology of the Broken Hill District. E. C. Andrew. (11) Memoirs Geol. Surv.: Geology, No. 8.

MECHANICAL.

Discharge Device for Fine-Ore Bins. H. H. Hunner. (6) Feb. 11, 1922.
M'Lellan Rock-Drill Bit. H. A. Kee. (6)
Leakage in Compressed-Air Lines. T. Simons. (6) March 18, 1922.
Mine Car Axle and Bearings. H. A. Sutliffe. (6) March 25, 1922.
Electricity Furnace for Drill Steel. T. M. Bains. (6) April 1, 1922.
Use of Slip Scrapers in Carson Hill Mines. F. S. Ferraris. (6) April 29, 1922.
Repairing a Compressor-Valve Cam. (6) May 6, 1922.
Device for Oiling Hoisting Rope. H. A. Linke. (6) May 13, 1922.
Measurement of Air. A. S. Fry. (8) Feb. 25, 1922.
Underground Loading Devices. A. E. Ring. (8) March 18, 1922.
Steam-Shovel Operation at Bisbee, Arizona. H. M. Ziesemer and G. Mieyr. (29) Feb., 1922.
Use of Wire Ropes in Mining Operations. J. F. Howe. (29) Feb., 1922.
Safety Practice for Hoisting Ropes. R. M. Raymond. (29) Feb., 1922.
Electric Haulage Systems in Butte Mines. C. D. Woodward. (29) Feb., 1922.
Stope Cost Records and Mine Contracts of the Anaconda Copper Mining Co. C. L. Berrien. (29) Feb., 1922.
Electrical Signal Installations in Butte Mines. C. D. Woodward. (29) Feb., 1922.
Power-Distributing Systems for Deep Metal Mines. C. D. Woodward. (29) Feb., 1922.

METALLURGICAL.

Some Essentials for Success in Pyritic Smelting. A. J. Bone. (6) Feb. 11, 1922.
Separating Platinum from Sand. (6) Feb. 11, 1922.
Chloride Volatilization. T. Varley. (6) Feb. 18, 1922.
Roasting Zinc Ores for Electrolytic Reduction. T. French. (6) Feb. 18, 1922.
Crushing-Surface Diagrams and Rittinger's Law. A. O. Gates. (6) Feb. 25, 1922.
Westley Electric Furnace for Copper Smelting. C. S. Wetherell and H. E. Skougor. (6) March 4, 1922
An Improvement in the Manufacture of Vessels Used in Retorting Zinc Ores. J. P. Varian. (6) March 4, 1922
The Power-Load Curves of Ball Mills. A. O. Gates. (6) March 25, 1922.
A Novel Cupelling Furnace. C. Flury. (6) March 25, 1922.
Cyanide Leaching of Copper Ores Impracticable. (6) April 1, 1922.
Lead Losses in Smelting Low-Grade Material. Dr. C. Offerhaus. (6) April 8, 1922.
Heat Requirements of Chloridizing Volatilization. O. C. Ralston. (6) April 15, 1922.
Asbestos Mining and Milling in Quebec. W. A. Rukeyser. (6) Serial commenced April 15, 1922.
Sulphatizing Roasting of Semi-Oxidized Copper Ores. P. R. Middleton. (6) April 15, 1922.
Mineral Sulphides as a Source of Electric Energy. G. N. Libby. (6) April 22, 1922.
Analytical Distillation of Shale Oils. M. J. Gavin. (6) April 22, 1922.
Heap Leaching of Low-Grade Copper Ores. J. Irving. (6) Serial commenced April 29, 1922.
Volumetric and Gravimetric Determination of Zinc in Ores. (6) May 6, 1922.
Non-Ferrous Metallurgy in Russia. M. G. Korsunsky. (6) May 13, 1922.
Checking Assay Samples by Screening Analysis. A. W. Allen. (6) May 13, 1922.

Refinery Practice and Flue Losses in Smelting Cyanide Precipitate. E. R. Richards. (6) May 20, 1922.
Cost of Milling and Treatment of Ores. (6) May 27, 1922.
Differential Flotation in Tasmania. (6) June 3, 1922.
Heat Requirements in Chloridizing Volatilization. R. Gahl. (6) June 3, 1922.
Cost of a Small Flotation Plant. E. Gayford. (8) Feb. 11, 1922.
Cyanide of Flotation Concentrate. P. T. Bruhl. (8) Feb. 11, 1922.
Treatment of Concentrate. E. P. Crawford. (8) Feb. 18, 1922.
The Function of Oil in Flotation. D. W. Leeke and R. H. Jarvis. (8) Feb. 18, 1922.
Thermal Requirements of Chloridizing Volatilization. H. R. Layng. (8) Feb. 25, 1922.
Hydro-Metallurgy of Low-Grade Zinc Ores. M. Namba. (8) March 4, 1922.
Wetting and Amalgamation. T. T. Read. (8) March 11, 1922.
Cyanide Regeneration. H. R. Layng. (8) March 18, 1922.
Artificial Pebbles for Tube-Milling. A. W. Allen. (8) March 25, 1922.
Gas Absorption and Oxidation of Non-Ferrous Metals. B. Woyski. (29) June, 1922.
Sampling and Estimating Disseminated Copper Deposits. J. B. Joralemon. (29) Feb., 1922.
Arsenical-Bearing Metals. H. J. Roast and C. F. Pascoe. (29) Feb., 1922.
The Treatment of Pyritic Concentrates Containing Tin. C. W. Gudgeon. (25) May, 1922.

MINING.

The Incline Slicing Method as Applied to Large Ore-Bodies. J. P. Hodgson and J. Kiddie. (6) May 27, 1922.
A Sub-level Slicer. W. H. Newett. (6) May 27, 1922.
Method of Sinking Drop Shaft in Soft Water-Bearing Soil. (6) May 27, 1922.
Efficiency in Use of Safety Fuse. G. H. Tod. (6) June 3, 1922.
Hanging Scaffold for Taking Down Loose Back in High Ground. J. H. Cornell. (6) June 3, 1922.
Mining Methods at the Ashio Copper Mine. M. Otagawa. (29) Feb., 1922.
Ventilation of Butte Mines of Anaconda Copper Mining Co. A. S. Richardson. (29) Feb., 1922.
Mine Fires and Hydraulic Filling. H. J. Rahilly. (29) Feb., 1922.
Efficient Ventilation of Metal Mines. D. Harrington. (29) Feb., 1922.
Spies' Open-Stope System of Mining. S. R. Elliott. (29) Feb., 1922.
Handling Ore in Mines of Butte District. H. R. Tunnell. (29) Feb., 1922.
Safety Devices for Mine Shafts. R. Kudlich. (29) Feb., 1922.

MISCELLANEOUS.

Crown Reef Mining and its Influence on Mining Costs. L. G. Hutchinson. (25) May, 1922.
Brown Coals of Victoria. H. Herman. (10) Geol. Surv. Bull. No. 45.
Inefficiency of Mine Labour. B. F. Tillson. (6) March 11, 1922.
Costs of American Copper Production, 1919–1920 (Inclusive). H. A. C. Jenison. (6) March 18, 1922.
Promoting Efficiency in Drilling and Blasting. B. F. Tillson. (6) March 18, 1922.
The Marketing of Pyrites. G. J. Young. (6) April 8, 1922.
Shaft-Sinking in a Limestone Quarry. J. H. Stovel. (6) April 8, 1922.
The Functions of Institutes. T. A. Rickard. (6) Serial commenced April 15, 1922.
Marketing Tungsten Ores. C. Hardy. (6) April 22, 1922.
Cobalt : its Past and Future. C. W. Knight. (6) May 6, 1922.
Marketing of Silver Bullion. P. D. Handy. (6) May 6, 1922.
Dredge and Power-Plant Construction in the Far East. P. R. Parker. (6) May 13, 1922.
Selection of Fans for Pipe Ventilation. W. S. Weeks. (6) May 13, 1922.
The American Zinc Institute. F. E. Wormer. (6) May 13, 1922.
Promoting Efficency in Drilling and Blasting. E. L. Watts. (6) May 20, 1922.
The Marketing of Arsenic. F. Y. Robinson. (6) May 20, 1922.
Sampling of Oil Shale. C. C. Star. (6) May 20, 1922.

Ajo Enterprise of the New Cornelia Copper Co. A. W. Allen. Serial commenced June 3, 1922.

Mining Instruction at Harvard. G. J. Young. (6) June 3, 1922.

Marketing of Talc. R. B. Ladoo. (6) June 3, 1922.

Brick-Making in Remote Mining Districts. H. C. Robson. (8) Feb. 11, 1922.

The Premium on Gold. S. J. Kidder. (8) Feb. 18, 1922.

Accident Prevention at the Calumet and Arizona Mines and Smelters. A. W. Allen. (8) Feb.

Ventilation and Human Efficiency. L. Hill. (8) Feb. 25, 1922.

Errors Latent in Mine Sampling. H. E. Black. (8) Feb. 25, 1922.

An Inventory of Results of Accident Prevention. C. A. Allen. (29) May, 1922.

Organization of Mine Sampling at Anaconda. W. B. Daly and F. A. Linforth. (29) Feb., 1922.

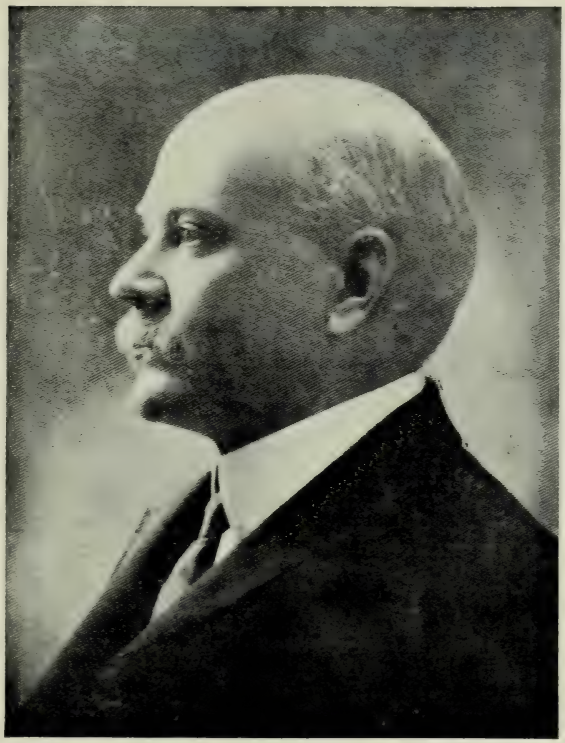
Sampling and Estimating Zinc and Lead Ore-Bodies in Mississippi Valley. W. F. Boericke. (29) Feb., 1922.

Bureau of Safety of Anaconda Copper Mining Co. C. W. Goodall and J. L. Boardman. (29) Feb., 1922.

The Re-establishment of the Gold Basis of Currency. H. R. Sleeman. (25) May, 1922.

Notes on the Valuation of Ores, Concentrates, and Smelter Products. L. C. Stuckey. (25) March, 1922.

Methods of Measuring Horizontal Angles Involving Steep Precipitous Sighting. L. H. Cooke. (25) March, 1922.



ROBERT CARL STICHT.

BORN 1856—DIED APRIL 30, 1922.

The death of Mr. Robert Sticht, general manager of the Mt. Lyell Mining and Railway Company, deprives Australian mining science of one of its most distinguished members. He was associated with Mt. Lyell from the time its resources began to be developed by the company. After an examination of the huge pyritic mass there, Mr. Sticht declared that he had never seen a deposit of ore that so lent itself to true pyritic smelting. Proceeding on that basis, he designed the equipment for the mine. Before Mr. Sticht's arrival in Tasmania from the United States his scientific attainments were widely recognised by metallurgists and mining men, but his reputation extended tremendously as years went on, and the value of his work at Mt. Lyell became known. Dr. Peters, the authority on copper smelting, when compiling the last edition of his book on that subject. stated that the sections dealing with the important matter of pyritic smelting were practically due to Mr. Sticht, to whom the proofs were sent for revision. During his long control of the Mt. Lyell mine and works Mr. Sticht never flagged in his investigations, and always encouraged the members of his staff to engage in original research. His discipline was strict, but had regard to the personality of the men under his control. As a judge of character and of capacity he was remarkably accurate—a quality that was probably responsible for the confidence and trust reposed in him by his workpeople. The atmosphere he created at the mine was reflected in Queenstown, and had as its reward the fact that although the company had differences with its employees these were fought out as matters into the settlement of which violence must not be imported. Mr. Sticht took the deepest interest in general science, literature and art. He was a Past President of the Australasian Institute of Mining and Metallurgy, a contributor to its publications, and a Member of Council at the time of his death.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED)

NEW SERIES.

No. 47.

27495
18.10.27

Edited by the Secretary.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

30th SEPTEMBER, 1922.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
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Australasian Institute of Mining and Metallurgy

(INCORPORATED).

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North Queensland	MURRAY RUSSELL, Department of Mines, Brisbane.
Papua - -	W. G. LANGFORD, Oilfields, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

COLLINS COURT, 374 LITTLE COLLINS STREET, MELBOURNE, VICTORIA.

Institute Matters.

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MINUTES OF MEETINGS.

OF THE INSTITUTE.

First Ordinary Meeting, 1922, Port Pirie, 25th to 30th September.

MONDAY, 25TH SEPTEMBER.

The First Ordinary Meeting, 1922, commenced at Port Pirie on Monday, 25th September. Members from the Eastern States, Tasmania, and Western Australia met on the previous day at Adelaide and journeyed to Port Pirie by train in reserved carriages. *En route* they were joined by members from Gawler and Moonta. On arrival at Port Pirie the train was met by Mr. H. St. J. Somerset (chairman), G. Dey (hon. secretary), and other members of the local committee. After being escorted to their hotels the visitors were accorded a civic reception in the town hall by the Mayor of Port Pirie. Mr. C. C. Freeman (Member of Council) acknowledged the compliment on behalf of the visitors.

In the evening a smoke concert was held in the town hall. This proved a very happy arrangement, providing as it did an opportunity for making members acquainted with one another before the more serious part of the meeting commenced. An excellent musical programme was carried out. This concert took the place of the usual Institute dinner.

TUESDAY, 26TH SEPTEMBER.

At 9.15 a.m. members assembled at the Library of the Broken Hill Associated Smelters. Here Mr. H. St. J. Somerset, superintendent of the smelters, had a model of the plant on view on a large table, and, alongside, another table showed a large flow-sheet of the operations. This flow-sheet carried samples of the products at the various stages of treatment. A typed brochure of points of interest was handed to each member (the contents of this brochure appear on pages xlv–lxv).

The party then divided into small sections, each under the leadership of a member of the B.H.A.S. staff, and the inspection proceeded throughout the day.

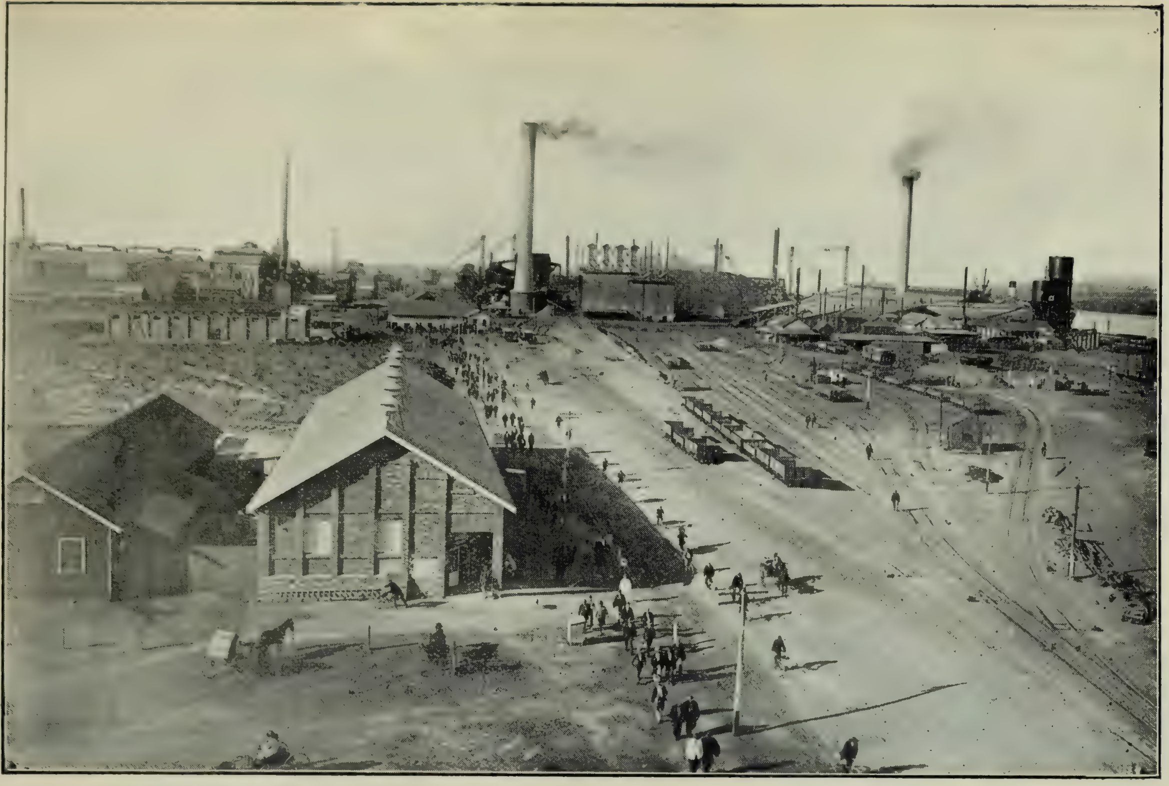
Guides during the inspection of the Works were:—

W. H. Campbell, Refinery Superintendent.
J. C. Hughes, Blast Furnace Superintendent.
J. Winter, Skinner Furnace and Pottery Superintendent.
P. J. Walsh, Dwight & Lloyd and Huntingdon & Heberlein Superintendent.
A. J. Hemingway, Chief Chemist.
G. Dey, Superintendent Experimental and Research Dept.
J. L. Sampson, Assistant Superintendent Refinery.
C. M. Rees Jones, Assistant Superintendent Blast Furnaces.
T. D'A. Melhuish, Assistant Superintendent Skinner Furnaces.
H. I. Campbell, Assistant Superintendent D. & L. and H. & H.
J. Hopkins, Assistant Superintendent Experimental and Research Dept.
J. Murrie, Experimental and Research Dept.

The following members of the staff also took part, although not attached to any particular party —

O. H. Woodward, Plant Superintendent.
S. E. Fraser, Chief Engineer.
R. N. Lewis, Power Engineer.
C. Mueller, Assistant Power Engineer.
J. Ahern, Plant Engineer.
T. R. Dibdin, Construction Engineer.
H. Collins, Electrical Engineer.

First Ordinary Meeting, 1922. Members at Co-operative Stores, Port Pirie.



The Broken Hill Associated Smelters Pty. Ltd.
View of Works taken from Flagpole, Harbor Master's Station, Port Pirie. 27/11/22.

Messrs. Gerald Mussen, Industrial Adviser; A. E. Davis, Secretary of the Co-operative Council, Sick and Accident Fund, and Provident Fund, and J. Retchford, Manager of the Co-operative Store, gave the visitors information on these matters.

In the evening a business meeting was held in the town hall.

Mr. C. C. Freeman presided.

Apologies for non-attendance were received from a number of members, including Messrs. G. C. Klug (President), Colin Fraser, W. E. Wainwright, A. S. Kenyon, H. A. Evans, J. M. Bridge, James Hebbard, Frank Moss, and Professor E. W. Skeats.

Minutes of the Annual Meeting, 1921, were read and confirmed.

An invitation from the Broken Hill Associated Smelters Co-operative Stores to visit the stores and make purchases was submitted, and accepted with thanks.

It was resolved that the fixing of the locality of the First Ordinary Meeting, 1923, be left to the discretion of the Council.

The Chairman expressed the thanks of the Institute to the Broken Hill Associated Smelters Ltd. for the privilege of inspecting the works and for the excellent provision made for the entertainment of the visitors. He also expressed the Council's appreciation of the work of the local committee in connection with the programme arrangements at Port Pirie.

On the motion of Mr. C. H. J. Clayton and Mr. F. Voss Smith it was resolved that the following recommendations be submitted to the Council:—

(1) "That indexing mining literature be again urged by the Council."

(2) "That the Council should see that the Institute is represented on all commissions, etc., dealing with the mining industry or any phase thereof."

(3) "That the Council should do its utmost to further the proposed Federal Geological Survey, as an aid to prospecting, and also consider whether the Federal Geological Survey should specialize along oil-prospecting lines."

(4) "That the Council be asked to approach the Federal Engineering Council with a request that they get into touch with the various authorities and urge that con-

cessions in fares be granted to members of the Institute travelling to meetings from the various States even if less than six (6) travel from any particular State."

The Chairman said that he did not propose to ask for discussion at that meeting on the various papers listed, most of which had been printed and supplied to members. During the inspection of the works members would have excellent opportunity for asking any questions on the subject of these papers.

The following papers which had not been printed were then read by the authors:—

"Blast Roasting Practice in the Treatment of Lead Ore at the Port Pirie Works of the Broken Hill Associated Smelters Pty. Ltd." By P. J. Walsh.

"Accounting and Costing." By A. Cottell.

"Construction Costs." By G. McI. Hunter.

On the motion of Mr. E. J. Horwood, a vote of thanks to the authors of all the papers was carried by acclamation.

The Chairman invited members to send in written discussion on the papers to the Secretary.

Mr. Somerset said that members interested in the working of the co-operative stores and of the Sick and Accident Fund would have these matters explained to them.

WEDNESDAY, 27TH SEPTEMBER.

During Wednesday inspection of the works was continued, and special arrangements made for any member to more minutely inspect any part in which he was particularly interested.

In the evening members were entertained at dinner at the Barrier Hotel by the Broken Hill Associated Smelters. Mr. H. St. J. Somerset presided. The dinner was excellently served, and a most enjoyable evening was spent.

After the loyal toast had been honoured, the Chairman gave the toast of "The Visitors." This toast was acknowledged by Messrs. C. H. J. Clayton (Broken Hill) and S. G. Turrell (Kalgoorlie) on behalf of the members and by Mr. Leo Cohn (Bendigo) on behalf of the students. "The Broken Hill Associated Smelters Ltd." was given by Mr. A. D. Greenlees (Broken Hill), supported by Mr. E. P. Grove

Whyalla Institute, built by B.H.P. Coy

Iron Knob Recreation Hall built and equipped with (Picture) Bioscope, Reading and Games Rooms, and Billiard Room (2 tables).

Members at Iron Knob.

(Melbourne) and Mr. J. O. Hudson (Hobart). Mr. M. Warren responded on behalf of the B.H.A.S. On behalf of members Mr. F. Voss Smith (Broken Hill) very cordially thanked Mr. George Dey, hon. secretary to the local Committee, for the work he had done in carrying out the arrangements for the local Committee, and asked Mr. Dey to accept a small present from members. Mr. Smith's remarks were supplemented by the secretary (Mr. D. L. Stirling) on behalf of the Council.

A vote of thanks to the Chairman was ably proposed by Mr. G. M'I. Hunter, and carried with enthusiasm.

The singing of "Auld Lang Syne" brought to a close a very pleasant evening.

THURSDAY, 28TH SEPTEMBER.

At 7 a.m. the steam tug *Uradlia*, belonging to the Adelaide Steam Tug Co. Ltd., was boarded, and after a delightfully calm passage of about three hours across Spencer Gulf Whyalla was reached. Here the party was met and welcomed by Mr. E. J. Horwood, of the Broken Hill Proprietary Co., and other members of the staff of that company. After an inspection of the power plants and the distilling plant, members travelled by the Broken Hill Proprietary Co.'s train to Iron Knob, a distance of 35 miles. After visiting the hall, library, and billiard rooms, provided by the company, members were the guests of the B.H.P. Co. at luncheon. An inspection was then made of the ore bins and haulage arrangements. An inspection of the quarry near the summit of the Iron Monarch followed. From the top of this iron mountain, which rises abruptly 620 ft. above a flat salt-bush plain, a very extensive panoramic view was obtained.

The party left for Whyalla at 4.30 p.m. Before embarking for Port Pirie a visit was paid to the Institute hall. This hall, which has a billiard room and reading room attached, was donated to the township by the B.H.P. Co. Port Pirie was reached at about 9.30 p.m.

A summary of the main points of interest at Whyalla and Iron Knob was prepared by the B.H.P. Co. and distributed amongst the visitors. This summary is published on pages lxvi–lxxiv of this number.

FRIDAY, 29TH SEPTEMBER.

Visiting members departed from Port Pirie for their homes by the morning and afternoon trains.

The following members of the staff of the B.H.P. Co. acted as guides during the inspection:—

E. J. Horwood, Superintendent of Mines.
H. P. Wright, Superintendent, Whyalla.
J. P. Williams, Electrical Engineer.
W. D. Wileman, Transport Officer.
J. A. Feist, Maintenance Engineer.
Frank Ralph, Assistant Electrical Engineer.
T. L. Barson, Assistant Engineer.
R. A. Clinton, Superintendent, Iron Knob.
J. Sly, Quarry Foreman.
H. J. Lugg, Under Foreman.

THE BROKEN HILL ASSOCIATED SMELTERS PROPRIETARY LIMITED, PORT PIRIE, S.A.

SUMMARY OF POINTS OF INTEREST

LEAD SMELTERY.

In the Flow Sheet of the Lead Smeltery (see Fig. 1) the numerals denote the various raw materials and products, both finished and unfinished, that are treated or produced at the Plant.

In the following sheets typical analyses and the destinations of these materials are given.

(1) LEAD CONCENTRATES (Average),

Delivered to D. & L. Plant, "A" Section.

SiO_2	4.6 %
FeO	4.0 %
MnO	2.5 %
CaO	0.69 %
MgO	0.19 %
Al_2O_3	1.4 %
Zn	9.7 %
S	16.1 %
S as SO_4	0.9 %
As	0.20 %
Sb	0.15 %
Pb	59.0 %
Ag	26 ozs.

(2) SLIME CONCENTRATES (Average),

Delivered to D. & L. Plant, "A" Section.

SiO_2	9.2 %
FeO	5.8 %
MnO	1.7 %
CaO	1.4 %
MgO	0.3 %
Al_2O_3	2.0 %
Zn	13.5 %
S	17.1 %
S as SO_4	1.17 %
As	0.08 %
Sb	0.19 %
Pb	43.2 %
Ag	45 ozs.

(3) LIME SAND.

(Forms part of D. & L. Charge.)

SiO_2	6.10 %
Fe_2O_3	0.77 %
Al_2O_3	0.66 %
CaO	47.00 = 83.93 % $CaCO_3$
MgO	3.50 = 7.35 % $MgCO_3$
Cl	0.067 %
SO_3	0.36 %
P_2O_5	0.157 %
H_2O	0.35 %
Organic Matter	0.19 %

(3A) LIMESTONE.

(Forms part of Blast Furnace Charge.)

SiO_2	6.80 %
Fe_2O_3	1.29 %
Al_2O_3	2.00 %
CaO	42.38 = 75.67 % $CaCO_3$
MgO	5.72 = 12.01 % $MgCO_3$
Cl	0.40 %
SO_3	0.25 %
P_2O_5	0.28 %
H_2O	0.75 %
Organic Matter	0.33 %

(4) CRUSHED IRONSTONE.

(Forms part of D. & L. Charge to "A" Machines.)

SiO_2	2.61 %
Fe_2O_3	91.31 %
Mn_3O_4	0.85 %
CaO	0.37 %
MgO	0.65 %
Al_2O_3	1.82 %
Cl	0.032 %
SO_3	0.12 %
CO_2	0.26 %
P_2O	0.106 %
Hygr. H_2O	0.55 %
Combined HO_2	1.28 %

(4A) IRONSTONE.

Analysis—see No. 4.

(Forms part of Blast Furnace Charge.)

(5) SILICIOUS ORE.

(Forms part of D. & L. Charge, "A" Section.)

SiO_2	39.44 %
FeO	8.24 %
MnO	4.11 %
CaO	1.86 %
MgO	0.35 %
Al_2O_3	7.09 %
Zn	5.5 %
S	4.1 %
S as SO_4	0.7 %
Cl	0.113 %
As	0.034 %
Sb	0.044 %
Pb	19.2 %
Ag	15.2 ozs.

(6) GRANULATED SLAG.

(Forms part of D. & L. Charge, "A" Section; also sent to Dump.)

SiO_2	20.7 %
FeO	28.0 %
MnO	4.5 %
CaO	11.6 %
MgO	2.0 %
Al_2O_3	6.2 %
ZnO	20.92 %
S	2.7 %
As	0.02 %
Sb	0.05 %

(6A) RETURN SLAG.

(See Analysis of Granulated Slag; forms part of Blast Furnace Charge.)

(7) D. & L. PRE-ROAST.

(Sent to "B" Section, D. & L. Plant, H. & H. Pots.)

Average S = 8.5 %.

(8) D. & L. SINTER.

(Sent to Blast Furnaces.)

Pb	45.01 %
Ag	24.5 ozs.
SiO_2	10.05 %
FeO	14.98 %
MnO	3.03 %
CaO	4.85 %
Zn	10.0 %
S	2.75 %
As	0.20 %
Sb	0.11 %

(9) H. & H. SINTER.

(Sent to Blast Furnaces.)

Pb	44.74 %
Ag	24.11 oz.
SiO_2	10.58 %
FeO	13.96 %
MnO	2.94 %
CaO	4.85 %
Zn	10.0 %
S	2.78 %
As	0.26 %
Sb	0.11 %

(10) COKE.

(Sent to Blast Furnaces.)

H_2O	0.7 %
V.H.C.	2.0 %
F.C.	80.1 %
Ash	17.2 %
S	0.45 %
Cl	0.018 %
Calorific power	11,894 B.T.U.

(11) FLUE DUST.

(Delivered to H. & H. Plant.)

Pb	42.5 %
Ag	9.0 ozs.

(12) SMELTER BULLION.

(Delivered to Refinery Copper Softeners.)

As	0.5 %
Sb	0.7–0.8 %
Ag	55–60 ozs.

(13) BAGHOUSE FUME.

(Sintered in Trucks.)

Pb	64.4 %
Zn	5.7 %]
Ag	1.2 ozs.

(13A) SINTERED BAGHOUSE FUME.

(Delivered direct to Blast Furnaces.)

Pb	64.4 %
Ag	1.2 ozs.
Zn	5.7 %
S	7.2 %
S as SO_4	2.3 %
Sb	0.42 %
As	0.6 %
Cd	4.1 %
Cl	6.83 %
Pb as $PbCl_2$	10.1 %

(14) COPPER DROSS.

(Sent to Blast Furnaces to recover Pb and Ag.)

Cu	10.83 %
Pb	72.20 %
Fe	0.90 %
As	2.47 %
Sb	0.83 %
S	5.10 %
Zn	2.81 %
Ag	30.0 ozs.

(15) ANTIMONY DROSS.

(Sent to Antimony Dross Furnace.)

Pb	71.5 %
Sb	8.9 %
As	8.3 %
Ag	0.5 ozs.

(16) ANTIMONY SLAG.

(Sent to Antimony Blast Furnace.)

Pb	56.88 %
Sb	15.91 %
As	11.60 %
SiO_2	1.67 %
Ag	0.4 ozs.

(17) WASTE SLAG: ANTIMONY BLAST FURNACE.

(Transported to Dump.)

0.5 % to 1.5 % Pb.

(18) ANTIMONY METAL.

(Tapped to Drossing Kettle and thence to Market.)

Pb	89.25 %
Sb	10.5 %
As	0.085 %
Zn	0.050 %
Ag	3.0 ozs.

(19) GOLD SKIMMINGS.

(Sent to Gold Alloy Furnace.)

Varying amounts of Au, Ag, Pb, and Zn; low in Ag.

(20) PRESSED SILVER ALLOY.

(Sent to Retort Furnaces.)

Varying amounts of Au, Pb, Ag, and Zn. 1500–2000 ozs. Ag per ton alloy.

(21) RETORT BULLION.

(Sent to Cupels.)

As	0.175 %
Sb	0.338 %
Zn	0.430 %
Ag	2180 ozs. per ton

(22) LITHARGE FROM SILVER CUPELS.

(Sent to Gold Alloy Furnace or to Blast Furnaces.)

Pb	89 %
Ag	150–300 ozs.

(23) FINE SILVER.

(Ready for Market.)

99.8 % Ag.

(24) REFINED LEAD.

(Ready for Market.)

Pb	99.988 %
Ag	0.00055 % = 3.6 dwt.
Bi	0.00107 %
As	trace
Sb	0.00713 %
Zn	0.00072 %

(25) LEAD REFINER DROSS.

(Sent to Blast Furnaces to Recover Pb and Ag.)

Pb	75.8 %
Zn	8.12 %
Sb	5.6 %
As	trace
SiO_2	0.45 %
Ag	0.46 ozs.

(26) HARD LEAD.

(Sent to Antimony Softeners to Recover Pb and Ag.)

Pb	96 %
Sb	1.5–2 %
As	1–2 %
Ag	9.0 ozs.

(27) METALLIC COPPER DROSS.

(Sent to Copper Softeners to Recover Pb and Ag.)

Cu	3.79 %
As	0.86 %
Sb	1.27 %
Zn	0.30 %
Ag	68.4 ozs.

(28) DROSS FROM MARKET KETTLES.

(Sent to Lead Refiners.)

Metallic Pb	60 %
Ag	0.1 oz.
PbO	Balance

Description of Dwight & Lloyd Sintering Plant.

The Plant is divided into five (5) sections :

1. Handling and mixing section to machine super hoppers.
2. " A " machines.
3. Intermediate crushing and conveying.
4. " B " machines and Huntington & Heberlein pots.
5. Delivery of product to smelter bins.

1. *Handling and Mixing Section.*—The flow sheet (Fig. 1) is self explanatory as regards the system of conveyors for handling the charge, etc. The mixing section has the following storage capacity :—

	Cub. ft.
Concentrates	1800
Slime concentrates	1830
Limesand	1100
Silicious ore	670
Granulated slag	1830

The charge mixture from each of the above bins is fed to the main collecting and mixing conveyor by geared driven belts which insure the desired proportion of each constituent of the charge being maintained at a constant figure.

2. " *A* " *Machines.*—This section consists of 4 D. & L. machines 252 in. x 24 in. Each machine has 42 pallets 42 in. wide x 24 in. long

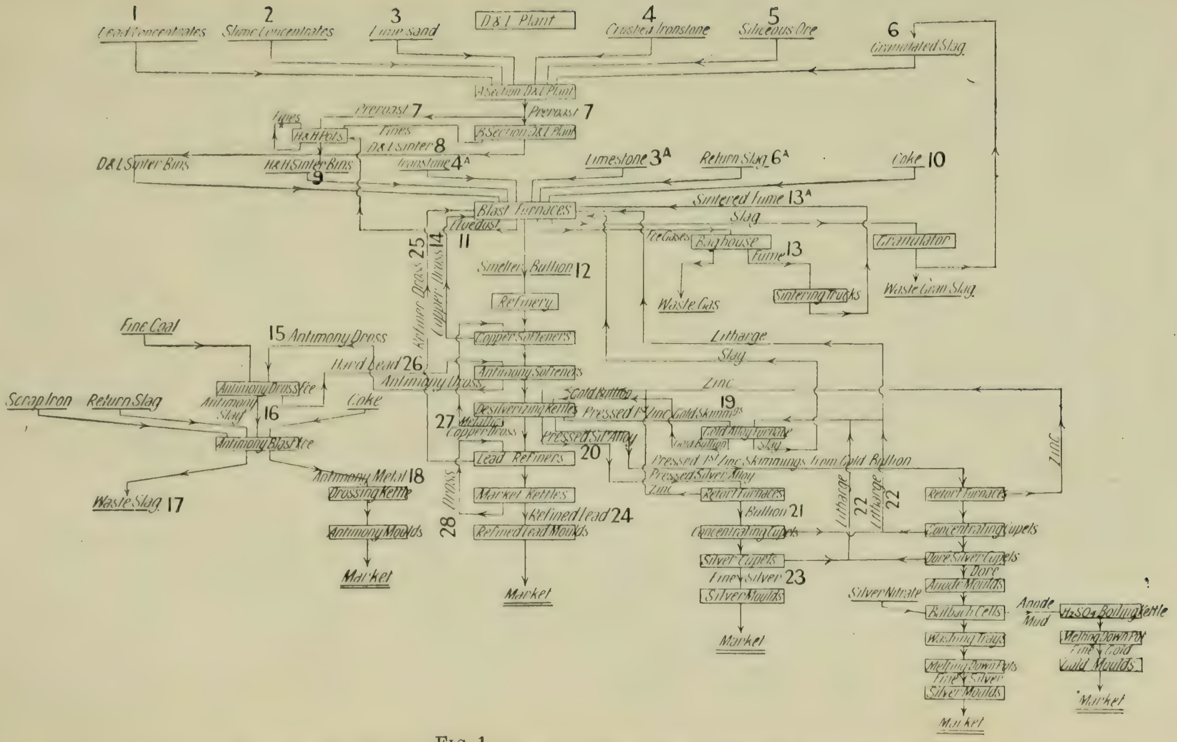


FIG 1.
FLOW SHEET—LEAD SMELTERY, BROKEN HILL ASSOCIATED SMELTERS LTD.

which travel over a vacuum chamber 21 ft. long. Grates of two types are used.

(*a*) Davison-Riddell Grates.—These are of the straight line type and have 41 apertures $\frac{10}{32}$ in. wide. Each pallet carries 3 of these grates which weigh 80 lb.

(*b*) Trail Grates.—These are built with 3 different types of bars so arranged that a loose bar alternates with a fixed bar. The loose bar rocks as the grates pass around the guide-rail and over the drive-wheel at the end of the machine, and thus the air spaces between the bars are kept open.

The pallet drop on the machines varies slightly but averages about 18 in. Each machine is fitted with a double inlet Kelly & Lewis fan running at 960 r.p.m. and driven with a 75 h.p. direct-coupled motor.

3. *Intermediate Crushing and Conveying Section.*—The preroast from " A " section falls from the end of the machine into tray conveyors at right angles to the discharge end of the machines. From the tray conveyors the preroast travels on belt conveyors to a bin, and thence through crackers to a pair of rolls which gave a product 84-85% through $\frac{1}{4}$-in. mesh. The discharge from the rolls then travels by belt conveyors to either the H. & H. Department or "B" section of the D. & L. plant.

4. " *B* " *Machine Section.*—This consists of 4 D. & L. machines which are similar to the " A " machines.

5. *Delivery to Smelters.*—The sintered material from the " B " machines falls direct into skips which are hoisted to the smelter bins by means of an aerial hoist.

METHOD OF WORKING.

Roasting is done in two stages :—

1. A preliminary or " A " roast done on D. & L. machines followed by a crushing of the " A " sinter.
2. A final or " B " roast of the " A " sinter on D. & L. machines or in H. & H. pots.

In both cases the charge is the same.

The raw charge from the super hopper of each machine is removed by a belt feed at the end of which is a spray to supply the requisite amount of moisture approximately 4-5%. The mixed damp material falls into an oscillating chute which distributes the charge lightly and evenly over the surface of the pallets. The heap at the back of the levelling plate is kept about 9 in. high in order to give a rilling action to the material. This causes the coarser particles to form the 1st layer on the grate and so prevents clogging. An adjustable plate fixed about 4½ in. above the grate fixes the depth of bed, and gives an uniform surface to the charge. The charge passes under a burner suspended over the end of the vacuum chamber. A coke fire with forced draught is used for ignition.

Vacuum on the chambers is—" A " Machines 9-14 in. W.G.
" B " ,, 3-6 in. W.G.

In the " B " machines the material is again moistened to a moisture content of about 5%.

Each preroast machine will treat approximately 200 tons of charge per 24 hours.

The following statement supplies brief data on the process :—

Preroast material averages	8.5% S.
Sintered material ,, ...	2.2% S.
	42.7% Pb.
	30.8 oz. Ag.
Speed of travel—" A " machines	24 in.-36 in. per min.
	17 in.-26 in. ,, ,,

Temperature of gases—100°-150° C.

% SO_2 in gases—0.6-2.0%, Average—1.4%.

Vacuum chambers are cleaned once in every four days.

Typical Charges.—

	Concentrate Charge. %	Slime Charge. %
Coarse Concentrates	42	24
Slime ,,	29	51
Limesand	5	4
Silicious Ore ..	2	2
Granulated Slag ..	10	11
Risdon Residues ..	12	8

Typical Sinters.—

	Concentrate Charge. %	Slime Charge. %
Pb	42.0	40.4
SiO_2 ..	9.1	10.1
S	2.3	2.2
Zn	13.3	12.0

The average Ag. in the sinter=33.8 oz. per ton.

Sampling Raw Materials.—Samples are taken at regular intervals at the plant during the 24 hours. These are mixed and cut down and from these daily and weekly composite samples are made up.

Sampling Sintered Products.—Samples are obtained by reserving regular pallets of the materials as the machine is operating and massing the samples of approximately 3 tons from all machines for a period of 8 hours. This is sent to the Sample Mill, crushed through ¼-in. mesh, and the sample mechanically separated.

HUNTINGTON-HEBERLEIN POT SECTION.

The H. & H. Pot Section works in parallel with the "B" section of the D. & L. plant. The charge consists of preroasted material from "A" section D. & L. The plant consists of 15 pots each 9½ tons capacity. For ignition purposes coke heated in a small furnace is used. Ignition coke per ton of charge=44 lb. The blast for sintering is supplied by the same turbo blower that supplies the blast furnaces. At the commencement of the operation the blast=4 oz. This is increased to 22 oz. at the end of the operation. Time required for sintering=7 hours. One attendant is required for every 3 pots. When sintering is complete the charge is tipped on to the floor beneath and then broken to a 6-in. gauge by contract labor. Fines (unsintered material) are returned to the pots. Sinter is loaded direct into skips and sent by aerial hoist to a 500-ton bin on the blast furnace feed floor.

The following is a comparison of D. & L. and H. & H. sinters produced from the same charge :—

——	To "A" Section. D. & L.	—	H. & H.	D. & L.
	%		%	%
Coarse Concentrates	24.0	Pb	42.5	40.4
Slime concentrates	51.0	SiO_2	9.4	10.1
Limesand	4.0	S	2.5	2.2
Silicious ore ..	2.0	Zn	12.0	13.3
Granulated slag ..	11.0	CaO	6.1	
Risdon residues ..	8.0			

The variation in the analysis of the H. & H. and D. & L. sinter is due to the fact that in the former process odd parcels of purchased lead-bearing material are fed direct to the pots and do not go through the preroast section of the D. & L. plant.

Blast Furnace Department.

Steam-driven hoists deliver the charge material to a series of 5 bins situated 13½ ft. above the furnace feed floor and running parallel to the blast furnaces. By means of a mechanical traverser the rake of trucks carrying the charge is transferred to a line running over the furnaces and by a similar arrangement the empties are returned to the refilling line under the bins.

Blast Furnace Charge.—

	lb.
D. & L. sinter	3000
H. & H. ,,	1000
Returned slag	1000
Coke	500
Lead dross	100

Blast Furnaces.—There are 5 large and 1 small furnaces all of which are connected with the main brick flue by means of downtakes.

The dimensions of the large furnaces are :—

Crucible inside measurements—Length 17 ft. 2 in. Width 4 ft. Depth 2 ft. 3 in.

They have the same dimensions at the tuyeres.

Water jackets are made of cast-iron with closed tops. Their dimensions are—Height 6 ft., thickness 6 in. with a 13-in. bosh on side and a 9-in. bosh on end jackets. There are 11 side jackets, fitted with 5 in. diam. tuyeres with cardboard slag escapes at the bottom, and three end jackets. At each end of the furnace is a small water-cooled tapping breast through which the slag is tapped. The shaft at the top of the jackets is 18 ft. 8 in. long x 6 ft. 2 in. wide, and at the feed floor 19 ft. 6 in. long x 7 ft. wide. From the bottom of the crucible to the feed floor the depth is 23 ft. 3 in. and from centre of tuyeres to feed floor 20 ft.

Flues and Downtakes.—On the large furnace the centre flue, which has an inverted V-shaped top of heavy iron plate dips into the shaft 12 in. below feed floor level and rises 6 ft. 6 in. above it. This flue extends the whole length of the furnace. It is 3 ft. wide leaving a space of 2 ft. on either side. From the central flue two downtakes each 4 ft. x 4 ft. take the furnace gases to the main flue which is on ground level. The furnace gases are delivered to the baghouse by means of a large suction fan running at 350 r.p.m. The heavy particles of flue-dust settle in the flue before reaching the baghouse. This material is sent to the H. & H. pots for treatment. The lighter particles are filtered from the gases by bags, and the fume recovered by shaking the bags. This fume readily ignites and burns to a comparatively solid clinker which is returned to the blast furnaces.

Composition of Sintered Baghouse Material.—

Pb	66.4%
$PbCl_2$	3.3%
Cl	5.0%
Zn	6.1%
Cd	3.3%
S	6.5%
S as SO_4	2.4%
Sb	0.08%
As	0.48%
Ag	1.1 oz.
Au	0.48 gr.

Blast.—This is supplied by a Fraser & Chalmers turbine blower. The normal working pressure is 54 oz. per square inch.

Smelter Bullion.—The bullion is drawn from the crucible through 4 firebrick syphons and moulded into bars weighing about 65 lb. and delivered to the refinery. The following is an analysis of the bullion :—

Sb	0.7–0.8%
As	0.2–0.4%
Cu	0.6–1.2%
Ag	55 –60 oz. per ton.

Slag.—The slag is tapped from the furnaces into a settling pot holding 0.75 tons. In this way lead suspended in the slag and also matte separate out. The slag overflows from a lip at the top of the pot and falls into a stream approximately 6 in. wide and 2½ in. deep into a cast-iron launder where it meets a spray of salt water. The water used includes water from the jackets.

From the launder the granulated slag is delivered into an elevator boot and thence to a 150-ton cylindrical bin. The slag is loaded from bins direct into side-tipping trucks and taken to the dump by a works locomotive.

Bucket Elevators for Granulated Slag.—End-drained buckets fixed on rubber belting.

Composition of Slag.—

SiO_2	20.0%
FeO	28.0%
MnO	4.0%
CaO	11.0%
MgO	2.0%
Al_2O_3	6.0%
ZnO	22.0%
S	2.7%
As	0.02%
Sb	0.05%
Ag	0.55 oz.
Cl	0.077%

Very little matte is made. The slag is generally hot and fluid giving very little trouble of any kind.

Production.—Average tons per furnace per day :—

Bullion	104 tons.
Charge	202 tons (excluding return slag and coke).
Slag	168 tons.

With 3 large and 1 small furnaces in operation the ratio between bullion produced and the amount of flue-dust and baghouse materials collected was :—

453 tons bullion per day.
4 tons baghouse material per day.
1.4 tons flue-dust per day.
Total slag fall .. 1.6 to 1.

Small Blast Furnace.—The dimensions are :—Crucible inside measurements—length 9 ft. 4 in., width 3 ft. 6 in., depth 1 ft. 9 in. Water jackets are 4 ft. 6 in. high, 6 in. thick with a bosh of $4\frac{1}{2}$ in, $3\frac{1}{2}$ in. dia. tuyeres on side jackets. Shaft at top of jackets 10 ft. 1 in. x 4 ft. 3 in. and at feed floor 10 ft. 6 in. x 5 ft. Height from bottom of crucible to feed floor 23 ft. 3 in. and from centre of tuyeres to feed floor 21 ft.

7 in. This furnace has an open top and is hand fed. The furnace is principally used for the treatment of refinery by-products such as process drosses and antimonial slag.

When treating antimony slag the following charge is used :—

	lb.
Antimony slag	1200
Returned slag	1000
Scrap iron	200
Coke	260

The products obtained are :—

(*a*) Antimonial lead.
(*b*) Slag and speiss which are tipped over the dump.

Refinery.

The base bullion from the blast furnaces is delivered to the refinery on horse-drawn trucks running on a 2 ft. 6 in. gauge track. Each truck carries 60 bars—approximately 2 tons. On arrival at the refinery it is weighed and elevated on the trucks by means of hydraulic lifts to the copper softening floor.

The refinery comprises 8 sets—3 large and 5 small. Each set is made up of the following :—

Furnace or Kettle.	No. of.
(*a*) Copper softening furnace	1
(*b*) Antimony ,, ,,	1
(*c*) Desilverising kettles	2
(*d*) Refining furnaces	1
(*e*) Market kettles	1

In addition there are the following adjuncts :—

Furnaces, etc.	No. of.
Antimony dross furnace	2
Gold alloy furnace	2
Retort distillation furnace	4
Cupel Furnaces :—	
Concentrating	7
Fine silver	3
Litharge making	1
Electrolytic parting plant	1

Copper Softeners.—The bullion is thrown by hand into the softeners through the working doors, the furnace being fired up to melt the bullion as it is being charged in.

The molten bullion is maintained at a temperature of 650-700° C., and the copper dross rises to the surface and is skimmed off by means of perforated skimmers, hand operated.

The dross—about 8% of the bullion treated—is returned to No. 7 blast furnace as described in the blast furnace section.

The analysis of the bullion after drossing is :—

As	0.4-0.6%
Sb	0.6-0.9%

Principal Data concerning the Copper Softeners.—

	Large.	Small.
Tank ..	22 ft. 4 in. x 13 ft. 4 in. x 21 ft. 10 in.	16 ft. 4 in. x 10 ft. 4 in. x 2 ft. 10 in. deep.
Lining ..	14 in. below jackets 9 in. above jackets	14 in. below jackets 9 in. above jackets
Hearth area ..	220 sq. ft.	112 sq. ft.
Grate area ..	31 sq. ft.	16 sq. ft.
Capacity ..	50 tons.	37 tons.
Coal consumption ..	2 tons per 24 hrs.	1.5 tons per 24 hrs.

The furnace is of the ordinary reverberatory type. The tank is lined with fire-brick, the bottom bricks being placed on a layer of a mixture composed of brick-dust, fire-clay, sand, and lime.

Antimony Softeners.—From the copper softeners the bullion is run off by means of cast-iron launders into the antimony softeners. The charge is maintained at a temperature of 950-1050° C.

The skimmings—oxides of As, Sb, and Pb—rise to the surface and run off through a notch in the end of the furnace.

The As and Sb in the softened bullion average :—

As	Nil
Sb	0.2-0.3%

The skimmings are retreated in the antimony dross furnace.

Description of the Furnace.—The furnace is similar to the copper drossing furnaces with the exception of water jackets ; these are necessary owing to the litharge corroding the furnace lining.

The centre jacket at one end of the furnace is provided with a V-shaped notch through which the skimmings run off.

Desilverizing.—The bullion from the softener is run down a cast-iron launder to the desilverizing kettle, which has been warmed and washed with lime water. There are two kettles to each softening furnace, as the operations in the kettle take twice as long as in the furnace. On tapping the charge into the kettle a quantity of dross is formed, which is skimmed and returned to the drossing furnace. The charge is then allowed to cool back to the solidifying point, when a crystalline metallic Pb-Cu alloy is thrown out and returned to the drossing furnace. The kettle is now fired strongly and the first, or gold, zincing is given. When the plates of zinc are melted the charge is stirred for 30 minutes, then cooled back and the alloy skimmed off and put aside. When the adjacent kettle is filled and ready for zincing this skimmed alloy is returned along with the Zn. The resultant alloy is put through the Howard press and treated in the gold alloy surface. The second, or first silver, zincing consists of a skimmed alloy off a previous third zincing or second silver zincing; for convenience these are called second and third zincings. The returns from a previous third zincing are put in the kettle, which is fired strongly; stirring commences when a piece of zinc thrown on the bullion melts readily, and is continued for 30 minutes. Cooling and skimming is repeated as before, the sides of the kettles being scraped down to remove adhering alloy. The alloy is pressed, broken into small pieces, and delivered to bins on the retort furnace floor. The third zincing is now given and the alloy put aside for the next second zincing. The handling of skimmings is done by pneumatic lifts. Provided the bullion does not exceed 4 dwt. Ag per ton after the third zincing, the charge is siphoned to the refining furnace, otherwise an extra zincing has to be given.

Refining Furnace.—The desilverized lead, containing 0.6% zinc and any impurities which escaped elimination in the previous operation, is siphoned to the refining furnace, which is a facsimile of the softening furnace. The zinc and other impurities are oxidized, forming a scoria which is removed into slag pots by rectangular perforated skimming plates and sent to the blast furnaces for recovery of the lead. The refining is complete when a sample of the lead poured into a warm mould and solidified is free of all evidence of zinc or antimony and the surface shows fern-like crystalline forms, the branches of which are at right angles to the axis.

Market Kettles.—From the refiner the lead is tapped to cast-iron market kettles, where, if still slightly impure, it is kept at a dull-red heat and a blast of air blown through it until a satisfactory sample is obtained, when it is cast into the various brands of lead according to sample and assay.

Brands of Lead Produced.—"Austral," "Special," "China." "Corroding."—Of better quality than "Special" and made by prolonging the refining process. "Superfine."—Of high purity, and is used as cathodes in electrolytic work.

Treatment of Antimonial Dross.—A typical sample of antimonial skimmings shows:—

Pb 69.1% Sb 8.8% As 7.2% Ag 2 oz.

This is mixed with slack coal in the proportion of 3000 lb. skimmings to 175 lb. of coal and charged into an ordinary reverberatory furnace. The hard lead, containing 2-3% Sb and 8-12 oz. Ag, is tapped from the reverberatory and returned to the softening furnace. The antimonial slag is run from the end of the furnace on to an iron plated floor, broken, and carted to the antimony blast furnace. The crude antimonial metal from this furnace is melted down in 40-ton lots in a kettle, carefully drossed to reduce the copper contents, and cast into bars. These bars contain:—

	%
Pb	85.25
Sb	14.5
Zn	0.24
As	0.85
Fe	0.14

Gold Alloy Furnace.—This is a reverberatory in all respects similar to a drossing furnace and is provided with a coal-fired kettle. This furnace is kept nearly full of bullion. About 5 tons of gold crusts are fed in each shift and strongly fired to oxidize the zinc, which escapes as fume or forms a mushy infusible dross. Litharge is then fed in, which scorifies the dross, rendering it sufficiently liquid to be raked into pots, and is returned to the blast furnaces.

The bullion is tapped into a kettle, moulded, and allowed to accumulate till sufficient to fill a zincing kettle. This bullion is then given a light zincing to extract as much gold and as little silver as possible. The resulting crust is removed, retorted, and cupelled to a Dore bullion, and sent to the parting plant. The bullion, after the zincing, is treated as ordinary bullion.

Retort Furnaces.—These are producer-gas fired regenerative furnaces. Each furnace accommodates four retorts, two on each side, and each side operates independently of the other. The retorts used are made by the Morgan Crucible Co. The weight of charge for a retort is 12 cwt. of zinc crusts, and the average life of a retort is 45 charges. A portable condenser is used, consisting of a wrought-iron cylinder lined with a mixture of sand, fire-clay, and cement. The zinc recovered from retorting is used on the desilverizing kettles. The dross is treated in cupels with retort bullion. The bullion is dipped from the retorts in iron ladles.

Cupellation Furnace.—There are seven furnaces used for concentrating the retort bullion to 16,000 oz. grade, 3 for running the concentrate to fine silver, and one for marketable litharge. The concentrating test consists of an elliptical iron basket of the following dimensions:—4 ft. 9 in. x 3 ft. 9 in. x 10 in. This basket is lined with the following mixture:—

Sand, 178 lb.	Cement, 717 lb.

A water-cooled pipe 1½ in. diam. fits neatly around the inside of the test at lead level. The litharge tapping breast is water-cooled. The silver cupellation furnaces are of the same dimensions, but the water-cooled circulating pipe is dispensed with. The litharge from these cupels is used on the gold alloy furnace. The silver is cast into bars of rectangular shape, 11½ in. x 5 in. x 5 in., and weighing between 1000 and 1050 oz. They are hammered while hot to knock down any excrescence and weighed on an Oertling balance. The litharge is sent to the gold alloy or antimony softening furnaces.

Parting Plant.—A Balbach electrolytic parting plant is used.

Zinc Roasting Plant.

SKINNER FURNACE.

In this plant zinc concentrates and zinc slimes from Broken Hill are roasted for the Electrolytic Zinc Co. to yield a calcine containing 6% S, which is then shipped to Tasmania for further treatment.

Material Treated.—

(*a*) Zinc Concentrates—

	De Bavay.	B. H. P. Co.
SiO2	2.1%	2.1%
FeO	10.7%	11.8%
MnO	2.4%	2.77%
CaO	0.9%	0.37%
MgO	Trace	0.13%
Al_2O_3	0.2%	0.2%
Zn	48.2%	47.00%
S	31.3%	29.50%
S as SO_4	0.37%	0.33%
Cu	0.4%	0.25%
As	0.2%	0.12%
Sb	0.13%	0.10%
Pb	6.0%	7.30%
Ag	10.0 oz.	13.0 oz.

(*b*) Zinc Slime Concentrates—

Zn	48.8%
Pb	3.0%
S	31.8%
Ag	11.0 oz.

Sizing Tests.—

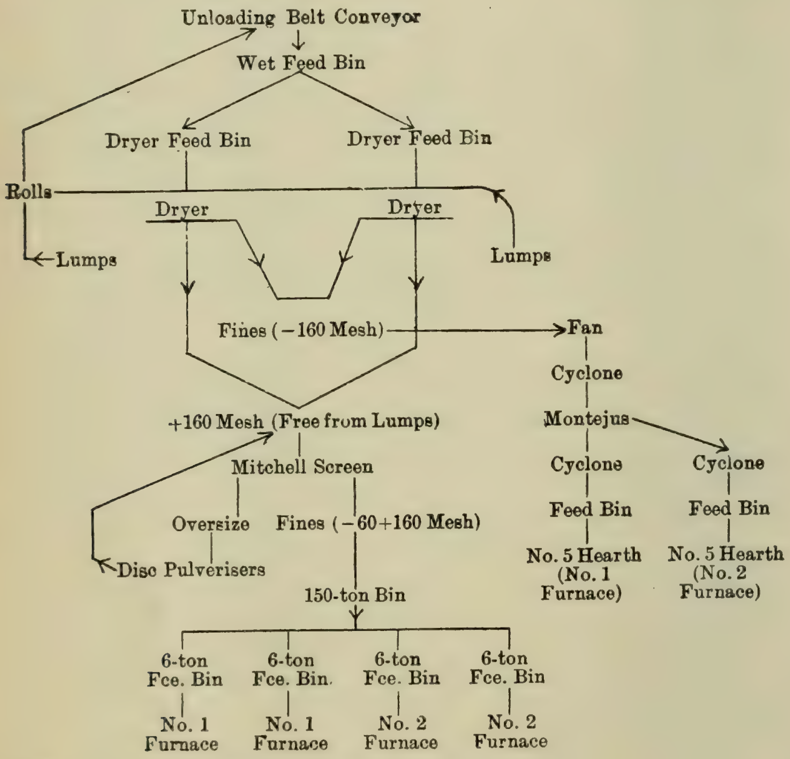
		(*a*) Concentrates. %	(*b*) Slimes. %
Representative Sample	+ 60 mesh	20	12.0
	+ 160 ,,	50	12.5
	− 160 ,,	30	75.5

Moisture.—The moisture content varies from 1-8 % but averages 5 %.

The Plant consists of:—

(*a*) Ore handling, dressing, and drying plant.
(*b*) The Skinner furnaces.
(*c*) Calcine handling plant.

(*a*) *Ore Handling, Dressing, and Drying Plant.*—The following flow sheet shows the passage of the material through this plant.



Thus the feed to the furnaces consists of :—

(1) Dry − 60 + 160 mesh material delivered to the top hearth of the furnaces.

(2) Dry − 160 mesh material delivered to No. 5 hearth of the furnace.

The Skinner Furnace.—Built of fire-brick blocks with steel buckstays and bonds.

No. 1 furnace consists of a dryer hearth and 8 roasting hearths.

No. 2 furnace consists of a dust-collecting hearth (No. 0) and 8 roasting hearths.

Each furnace has the following dimensions—

Height over all	36 ft. 1½ in.
Distance ground level to bottom hearth	6 ft. 6 in.
Diameter of furnace	22 ft. 2 in. ext. 20 ft. 0 in. int.
Diameter of central shaft	3 ft. 8 in.
Central dropholes	6 ft. 6 in. outside diameter 3 ft. 8 in. inside ,,
Outer dropholes—Semicircular	11 in. radius.
Ore travel in the furnace	72 ft.

Note.—In No. 2 furnace there are 18 outer dropholes in the outward rabbling hearths instead of 8 as in No. 1 furnace.

Also in No. 1 furnace there are 4 muffle hearths under hearths 3, 4, 5, and 7. In No. 2 furnace there are no muffle hearths.

Rabbling is done by two air-cooled arms in each hearth, these arms fitting into sockets in a central rotating column. Each arm carries 9 cast-iron removable teeth, these teeth being of two types :—

(1) Inner rabbling, e.g., in hearths 1, 3, 5, and 7.
(2) Outer ,, e.g., ,, ,, 2, 4, 6, and 8.

The ore travels alternately from the circumference to the centre and from the centre to the circumference on successive hearths. Communication between hearths is by means of dropholes, there being one central drophole on the inner rabbling hearths and 8 outer dropholes on the outward rabbling hearths. The calcine is finally discharged by an automatic self-sealing discharge door in No. 8 hearth. The arms and columns are air-cooled. A central rib in the column fits against a central fin in each arm and enables cold air to be blown up one side of the column through the arms and out to atmosphere or to the furnace hearths through the other half of the column. The temperature in the lower hearths is maintained by coal or oil fires. The upper hearths are cooled when necessary by opening the furnace doors, or by blowing in cold air from the blower supplying air to the column. The column is turned by a driving pinion and crown wheel, and is supported on roller bearings. The column drive is from 5 h.p. motor or belt coupled to gears and thence to the driving pinion.

Furnace Draught.—The furnace gases escape from the top hearth by a central offtake, thence through a cyclone to a brickdust chamber and finally to the stack. It is also arranged that a fan can be put in the circuit between the cyclone and dust chamber in the event of the pull from the stack not being sufficient. The supply of air to the furnace is regulated as follows :—

Preheated air (from the column) is supplied to hearths 4, 6, and 8 by means of a flue down the side of the furnace. Preheated air also enters No. 8 hearth through the fire-box. Cold air is admitted to the top hearths (*e.g.*, Nos. 2 and 3) by the furnace doors or by an air main which connects with the column air blower. The supply of air to the hearths is regulated entirely by the temperature required. The temperature required is 800°-870° C. Above 870° C. heavy lead losses occur and insoluble zinc compounds form. Below 800° C. hearth accretions, due to the formation of $ZnSO_4$, give considerable trouble. The hearths are barred regularly through the doors and through openings in the panels. The following are typical figures of the furnace operations :—

Temperature of air leaving column ..	240° C.
Tonnage treated	60 tons per day.
Coal used	2 tons per day.
SO_2 content of furnace gases	3.5%-4%
Sulphur in concentrates	31%
Sulphur in calcine	6%
S as SO_4 in calcine	1%-2%

The sulphur elimination on the various hearths is shown in the following table :—

		%
Off No. 1 hearth	..	28.29 S
,, ,, 2 ,,	..	23.59
,, ,, 3 ,,	..	18.22
,, ,, 4 ,,	..	13.51
,, ,, 5 ,,	..	10.85
,, ,, 6 ,,	..	8.85
,, ,, 7 ,,	..	6.75
,, ,, 8 ,,	..	4.65

Calcine Handling.—The hot calcine drops from No. 8 hearth through a chute into a hollow cylinder. This cylinder (14 ft. long x 2 ft. diam.) is set at an angle of 8° to the horizontal and turns at 8 r. p. m. The calcine enters the cylinder every minute in charges of 100-150 lb. In passing through the cylinder it is cooled, and emerges as a regular feed, which falls into the push conveyor. This conveyor carries the calcine to a cross conveyor, thence to an elevator and the calcine bin. From this bin it is loaded by a belt conveyor into railway trucks.

The calcine is moistened at the end of the push conveyor by means of water sprays, and further moistened on the belt which feeds the railway trucks. A moisture content of 8–10 % is desired in the calcine. The calcine is trucked to the wharf and shipped to the Electrolytic Zinc Co., at Risdon, Tasmania.

THE BROKEN HILL PROPRIETARY COMPANY LIMITED.

GENERAL DESCRIPTIONS OF QUARRIES AT IRON KNOB AND WORKS AT WHYALLA.

THE history of Iron Knob may be read in "From Silver to Steel," by Roy Bridges, procurable from all leading booksellers.

The geology of Iron Knob and Iron Monarch deposits is given in "Geological Survey of South Australia," by R. Lockhart Jack, B.E., F.G.S., deputy Government geologist. The iron ore deposits and the associated rocks form a ridge a mile and a half in length, rising to about 620 ft. above the surrounding plain. The southern hill is the Iron Monarch, and the northern hill Iron Knob. The central portion of the ridge is a ferruginous quartzite of no value as ore.

Quarry "A" is situated on the southern slope of the Iron Monarch.
Quarry "B" ,, ,, ,, ,, eastern ,, ,, ,, ,, ,,
Quarry "C" ,, ,, ,, ,, northern ,, ,, ,, Iron Knob.
Quarry "D" ,, ,, ,, ,, eastern ,, ,, ,, ,, ,,

The last-named quarry has not been worked for years.

Quarrying of the Ore.—The quarrying of the ore may be divided into four sections—

(1) Drilling of the ore.
(2) Blasting.
(3) Loading into quarry trucks.
(4) Transporting to bins.

(1) *Drilling.*—The drilling was, in the early stage, done with the hammer and drill; it is now carried out with compressed-air drills. Several types of machines have been tried out over long periods, and finally it has been decided that the best machines for the work are—

(*a*) The Clipper, manufactured by the Denver Rock Drilling Co., for deep holes.
(*b*) The Jackhammer, manufactured by the Ingersoll Rand Co., for popping.

The depth bored in hematite ore with the Clipper varies from 20 ft. to 50 ft. per shift, while in manganiferous ore from 2 ft. to 10 ft. is about the average. One man can operate this machine up to 12 ft. steel; for over this length two men are employed.

The Jackhammer averages from 20 ft. to 50 ft. per shift in hematite, and from 1 ft. to 10 ft. in manganiferous ore. This is a one-man machine. The hardness of manganiferous ore varies with the manganese content, and as a general rule the higher the manganese per cent. the harder the rock. The maximum depth of face hole bored is 21 ft. The holes in boulders for popping vary from 6 in. to 3 ft., according to the size of the boulder to be broken. The spacing and depth of the holes on top of the face end on the benches depends on the contour of the face where it is to be blasted. A 21-ft. hole would have a burden of from 20 ft. to 25 ft. on the toe of the hole. Holes spaced for simultaneous firing (electric blasting) would have a burden of from 20 ft. to 25 ft., and spaced about 35 ft. apart.

The nature of the rock and the local heads on the particular part of the face to be blasted governs the spacing of the hole and the explosives charge to a great extent. The steel used is 1-in. hollow hexagon. In the first stages of air rock-drills operation at Iron Knob, and when the E 102 Sargeant drills (piston drills) were under test, 1 in. to 1⅛ in. solid steel, manufactured at the Newcastle Steel Works, was used, and found to give quite as good results as the imported solid steel.

The Clipper (hammer) drills with hollow steel were, however, proved to give much better operation, and the Sargeant drills were discarded.

The bit used on the drills is the standard 4-point bit sharpened by pneumatic Leyner drill sharpener (Ingersoll Rand).

The pitching drills about 2 ft. long have a 2¾ in. bit, and a 21-ft. steel would have a 1.9/16 in. bit. The difference in length of each succeeding drill is 1 ft.

Compressed air is generated in the Iron Knob power plant. The compressor is of the condensing horizontal Corliss type, and capable of 2000 cub. ft. free air per min. at 72 r.p.m. The air is compressed to 100 lb. per sq. in. pressure. The boiler plant is of the standard B. and W. boilers.

(2) *Blasting.*—When the necessary holes have been drilled the hole is "chambered" or "bulled," using 50% gelignite. The charge of 25% ligdyn or black powder is then placed, and the primer, of gelignite, with fuse and detonator attached for a ligdyn charge or powder fuse for powder charge, is placed, and the hole tamped with sifted earth. For simultaneous firing electric detonators are used, connected in series, and then connected to an exploder. All firing is done by a powder gang after the quarrymen have ceased work, either at dinner time or in the evenings. When the holes have been

fired the powder gang goes over the face and makes it safe for the loading hand to work under. The ore is generally in too large lumps to be spawled, in which case it is blistered or popped. The latter whenever possible, as it is very much less expensive. Either gelignite or ligdyn is used for blasting or popping. Experimenting with explosives on a working scale is continuous, as it is in this way greatest efficiency is attained.

(3) *Loading into Quarry Trucks.*—This is either done by contract hand-shovelling, when the men work in pairs, averaging about 30 tons per man per day, or by steam shovels, averaging about 300 tons per shovel per day. Two steam shovels, Bucyrus 35 B, with 1 to $1\frac{1}{4}$-cub. yd. dippers, are used in quarry "A." It is later hoped that all loading will be done with power shovels. The normal weight of ore in a quarry truck is 5 tons. The maximum size lump loaded must pass a 12-in. ring.

(4) *Transporting to Bins.*—The loaded trucks are horse-drawn to the brace of the incline, hooked on to the incline rope, and let down to bins. In quarry "A" a new lay-out of trackwork is in course of construction. Locomotive traction will be later on employed in this quarry. The loaded trucks going down haul the empties up. Quarry "A" is served with two inclines—one incline (No. 1) running to a junction plat and thence down another incline (No. 3) to the bins; the other incline (No. 5) goes direct to another bin. On this incline the trucks are 12-ton capacity, and are not detached from the incline ropes, the brace being so constructed that quarry waggons tip into incline wagons. This direct incline is the latest constructed, as is also the bin that it serves. Quarry "B" output goes down an incline (No. 2) to the junction plat mentioned above, and then to the bins via No. 3 incline. Quarry "C" output is let down by No. 4 incline direct to bin.

Transportation of Ore from Iron Knob to Whyalla.—The ore is loaded from the bins above referred to into 15-ton capacity steel hopper trucks, hauled 34 miles to Whyalla, and tipped into crusher bins. Four locomotives (Baldwin)—two of 70 tons weight and two of 138 tons in working trim (the latter are the heaviest locomotives in Australia)—are available for main-line work. The 70-ton locomotives haul up to 400 tons of ore per trip, and the 138-ton locomotives haul up to 650 tons of ore per trip. One 40-ton locomotive is used for shunting at Iron Knob, and another at Whyalla. Westinghouse compressed-air brake equipment is fitted to all rolling stock. The permanent way rails are 80 lb. per yard section, rolled at Newcastle Steel Works.

Crushing of Ore.—From the crusher bin the ore is fed over grizzleys into five 30-in. x 18-in. Hadfield jaw crushers, crushing from 12 in. down to 3½ in. to 4 in. The crushed ore is delivered on to a 36-in. conveyor belt, then on to a 36-in. cross conveyor, and, lastly, on to the 26-in. storage bin conveyor, and tipped into the storage bin by means of a tripper, when the ore is then ready for shipment.

Loading Ore into Steamers.—Running in a gullet under the storage bin and along the jetty, 2800 ft. long, are a series of 36-in. conveyor belts, which convey the ore to the end of the jetty, over a boom, into a chute, and into the steamer's hold. The boom is used to run the chute in or out to trim the ship and to adapt itself to the beam of the steamer loading. There are seven conveyor belts, excluding the boom, along the length of the jetty—the first six operated by 75-h.p. motors, No. 7 by 110-h.p. motor. At present each motor, during loading, is attended by a motor attendant on the simple control system. In the course of a few weeks automatic control, now being installed, will be put into operation. A description of the control follows later under "Power Distribution."

Power Plant.—The power supply is alternating current at 525 volt 50 cycles. At present power is generated in the old station near the shore end of the jetty. A new and modern power plant is in course of construction, and will, in the course of a few weeks, be available for duty. A description of the new station follows later.

Water Supply.—Owing to the uncertain rainfall, fresh water is distilled in evaporating plants from the sea water to augment the supply from an 8,000,000-gal. dam at Whyalla and a 30,000,000-gal. Government dam at Iron Knob. A modern distilling plant is described later.

Workshops.—All general maintenance work, construction work, erection, and repairs to locomotives, etc., for Iron Knob and Whyalla is carried out in the Whyalla workshops.

Assay Office.—The ore is sampled and carefully assayed, so that a continuous record of the value of the ore quarried may be available. A complete knowledge of the ground ahead of the working face is made available by the use of a diamond drill.

General.—Accommodation for the workmen was, in the early years, timber and iron construction, reinforced concrete now being used for all cottage construction. A fine institute hall, built by the company and handed over to the people of Whyalla, is available for cinema shows and general entertainment. A similar hall is also in use at Iron Knob. Bathing and clothes-washing conveniences have been built by the company, and are available to the workmen at Whyalla and Iron Knob. Accidents and illness are taken care of by the company's doctor.

Description of Distilling Plant at Whyalla.

The plant is of the horizontal type, and was made by Messrs. Mirrlees Watson, of Glasgow, Scotland. Its capacity is 78,000 gal. of distilled water (net) per 24 hours.

The plant is designed on the multiple system of evaporation, and consists of six distilling vessels, with air and auxiliary pumps, surface condensers, and preliminary heaters. Each distilling vessel consists of a cylindrical shell containing two sets of tubes. The bottom, or smaller, set of tubes is for the pre-heating of the incoming salt water, and the larger, or top, set is for evaporating the salt water. The operation is as follows:—The salt water for circulating through the surface condenser is obtained from the near-by salt water ducts. The water required for evaporation is taken from the outlet of the condenser and pumped through No. 1 and No. 2 preliminary heaters, and then through the pre-heating tubes of the six evaporating vessels, starting at the low-pressure vessel and then going through each of the vessels in turn. On passing through the pre-heating tubes of the high pressure or No. 1 evaporating vessel the salt water passes into the evaporating coils of that vessel, where some of it evaporates, the remainder flowing into the evaporating coils of No. 2 vessel, and so on throughout the six vessels. In this way the salt water being fed to the plant is heated up in gradual stages until the temperature required for the evaporation in No. 1 vessel is reached, thereby reducing temperature strains and tube leaks to a minimum. Steam is supplied from the power plant boilers to the shell of No. 1 distilling vessel, and condenses there, causing the salt water to boil. The resulting vapour passes first into the separator, where entrained salt water is removed, and then into the shell of No. 2 vessel, where it condenses, giving up its heat to the salt water flowing in the evaporating tubes of No. 2 vessel. The vapour from No. 2 vessel similarly passes into the shell of No. 3 vessel and condenses there, and so on until No. 6 vessel is reached. The vapour from No. 6 distilling vessel, after passing through the separator, flows through No. 1 preliminary heater (thereby raising the temperature of the incoming salt feed), and then into the condenser.

As the evaporation proceeds throughout the six vessels the pressure falls in stages from 40 lb. in No. 1 vessel to 27 in. vacuum in the sixth vessel. In this way the latent heat of steam is utilized for distilling purposes six times before the condenser temperature is reached.

The live steam from the boilers, on condensing in the shell of No. 1 distilling vessel, is returned to the boilers, but the vapour from No. 1 distilling vessel, on condensing in the shell of No. 2 distilling vessel, passes into the shell of No. 3 vessel, where it gives up its temperature heat.

The vapour from No. 2 vessel, on condensing in the shell of No. 3 vessel, passes into the shell of No. 4 vessel, together with the condensed vapour from No. 1 vessel, and so on until finally the condensate from all five vessels is withdrawn through No. 2 heater into the condenser. In passing through No. 2 heater the condensate further raises the temperature of the incoming salt feed.

As the salt water passes through the various evaporating coils concentration takes place, until a heavy brine is obtained at No. 6 vessel. This brine is drawn off by means of the brine pump, and is discharged to waste.

The air pump, brine pump, salt feed pump, and distillate pump are all driven by the same engine. The circulating water is handled by a separate duplex steam pump.

These pumping engines receive steam at 100 lb. sq. in. pressure. and exhaust against a back pressure of 40 lb. into an oil separator. The exhaust steam is then admitted into the shell of No. 1 vessel, thereby reducing the live steam consumption by utilizing the latent heat of the exhaust steam.

The "make" of the distilling plant, on being withdrawn from the condenser by the air pump, is ready for use for steam-raising or domestic purposes.

NEW POWER PLANT AND POWER DISTRIBUTION SYSTEMS AT WHYALLA.

BOILER PLANT.

The coal for the above plant is dumped into the crusher hopper from bottom dump trucks, and crushed to the required size. It is then elevated and conveyed into the overhead coal bins, for which the supply of coal is drawn for five B. & W. boilers, which are fired by chain grate stokers. The boilers are of the double-drum type, each capable of evaporating 8700 lb. of water per hour at a pressure of 200 lb. per sq. in., and superheating the steam to 100° F. Five Green's economizers of 128 tubes each, arranged on the unit system (that is, one to each boiler). Each economizer is operated by its own motor. The boilers are operated on induced draught. Two engine-driven turbon fans (either of which is of sufficient capacity for the entire boiler plant) provide the draught. The speed of the fans is automatically controlled by boiler pressure. Water is fed to the boilers by Weir type feed-pumps. Steam and feed water mains are designed on the ring system to safeguard the plant against shut-downs due to pipe failures. Ashes and soot are dumped directly from the boilers into hopper trucks running on suitably arranged lines.

ENGINE HOUSE.

1250 *K.V.A. Turbo Generator.*—The turbine is of the Rateau high-pressure impulse type, capable of driving the generator at a speed of 3000 r.p.m. when supplied with steam at 200 lb. per sq. in., superheated 100° F., and exhausting into a vacuum of 27.25 in. The turbine is controlled by a speed governor through the medium of oil relays. An independent automatic over-speed governor is also provided. Lubrication throughout the set is under pressure, and controlled by a pump driven by a worm off the turbine shaft. An auxiliary steam-driven oil pump is provided for starting up. The turbine exhausts into a Kelly & Lewis condenser set, consisting of a surface condenser and kinetic air pump and motor-driven circulating water pump. Three-phase 50-cycles current is generated at 525 volts. The generator is protected by a Merz–Price circulating current protective device.

60 *kw. Combination Separate Excitation and Auxiliary Lighting Set.*—This set is being installed, and consists of a 60 kw. steam

turbine, a 60 kw. A.C. generator, and a 35 kw. D.C. generator, all direct-coupled on one bed-plate. Under normal circumstances this set would be run up by the steam end (turbine) to synchronous speed. The A.C. unit would then be closed in to the A.C. busbars, and run as a synchronous motor. The D.C. generator is then closed into D.C. busbars to provide the station excitation, and the load is thrown on to the synchronous motor by adjusting the turbine governor to a position corresponding to a speed a few revolutions below synchronous. In the event of the works being shut down—for example, on such occasions as Sundays and holidays—electric lighting would still be required for the works and private consumers. This condition will be covered by running this set as a lighting set only, the turbine being the motive power, and exhausting into a feed heater.

Circulating Water.—The salt-water pumps are motor driven, and situated on the jetty at a point about half a mile from the power station. Cast-iron piping conveys the water to a duct of suitable capacity immediately outside the power plant. From this duct the various condensing sets will be supplied with circulating water through the medium of their own motor-driven circulating water pumps. The outlet from the condenser delivers into a suitable outlet duct, which returns the water to the sea. In the event of the failure of the jetty pumps, the ducts are of sufficient capacity to maintain the station until the stand-by jetty pumps can be started.

Hot-well.—The hot-well is arranged so that the condensate from any one unit or from the entire station can be measured as required.

Switchboard.—The switchboard is made up of a generator panel for each set, a separate excitation panel, a summation panel, a voltage regulation panel, and four feeder panels, all of which are equipped with the necessary indicating and recording instruments. The busbars and switches are contained in a fireproof chamber in the basement below the switchboard. The plant throughout has been designed with a view to extension. For this reason flues, ducts, piping, etc., are all of ample size, and the building is provided with an easily removable temporary end.

Distribution.—Current not being consumed in the immediate vicinity of the power station is stepped up by static transformers from 525 volts to 5250 volts, and is transmitted to two sub-stations, one situated at the crushing plant, the other in the middle of the loading jetty. At these two stations the voltage is stepped down from 5250 volts to feed 500-volt motors.

Control of Jetty Motors.—In order to obviate lost time when loading a ship, it has been the custom to employ one man to attend to the stopping and starting of each motor and its belt. Automatic equipment is now being installed which will put the control of all loading motors into the hands of one man, and will ensure that all the conveyor belts are started and stopped in a minimum of time. Each motor is provided with a contactor panel for starting and stopping. All of these panels are interlocked in such a manner that they will both start and stop the motors in the desired sequence. The position of the panel for operating this control is at the loading berth, immediately alongside the vessel being loaded. The equipment of this panel is three push buttons only. The first button controls the sequence of starting the motors, the second the sequence of stopping the motors, and the third is an emergency push button, and stops all motors immediately. In addition to this control panel, there are a number of emergency push buttons situated at convenient places along the entire length of the jetty. In the event of necessity arising for the loading to be immediately stopped, this can be done instantly by pressing any one of these buttons.

OF THE EXECUTIVE COMMITTEE.

JULY 10TH, 1922—1 P.M. (Special Meeting).

Resolved that the Council of the Australasian Institute of Mining and Metallurgy suggest that legislation be enacted in the various States for the following purposes:—

(1) To ensure that the prospectuses of every new mining or metallurgical enterprise intended for public subscription shall be supported by a synopsis of the latest report (if any) by the State Geological Survey or Mines Department on the area, such synopsis to be previously approved by the Department concerned.

(2) To render it obligatory that persons making technical reports for the promotion of mining companies should state in the report their degree or other qualifications, and should state the University or other institution from which they obtained such qualification.

(3) To urge the immediate necessity for a mining companies' law, especially relating to prospectuses, and providing that Directors shall be personally liable for any wilfully misleading statements in such prospectuses.

Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power were nominated to represent the Institute on the main committee of the Australian Engineering Standards Association.

AUGUST 14TH, 1922—1 P.M.

The secretary's report was presented, and accounts to the amount of £250, including printing, £90, line blocks and lithos., £15, bookcases, £77, salary and accommodation (2 months), £50, postage and sundries, £15, were passed for payment.

Nominations were submitted and dealt with as follow:—As Members—Edward William Finch and Percival Faraday Thompson (approved), Ernest F. Harris (deferred). As Associate Member—James Bain Auld (approved). Henry J. Truscott was admitted as an Associate Member, and Wm. Edward Cerutty as a Student.

The following papers were submitted and referred to the Publication Committee:—" Methods and Apparatus for the Determination of

Dust Suspended in Air," by P. H. Warren and T. A. Read; "The Gold-Antimony Veins of Costerfield," by Dr. F. L. Stillwell; "Notes on the Estimation of Tungsten in Ores," by J. R. Pound; "Volumetric Determination of Iron," by P. F. Thompson; "Blast-Roasting Practice in the Treatment of Lead Ore at the Port Pirie Works of the B.H.A.S.," by P. J. Walsh; "Lead Smelting Practice at the Port Pirie Works of the B.H.A.S.," by O. H. Woodward and J. C. Hughes; "The Practice of Desilverizing Base Bullion at the Port Pirie Works of the B.H.A.S.," by W. H. Campbell; "Analysis of Refined Lead as Practised at the Port Pirie Works of the B.H.A.S.," by A. J. Hemingway; "The Determination of Dust Losses in Metallurgical Processes as Practised at the Port Pirie Works of the B.H.A.S.," by A. J. Hemingway; "Industrial Assaying and a System of Costing as Practised at the Port Pirie Works of the B.H.A.S.," by A. J. Hemingway; "The Technical Analysis of Lead Blast-Furnace Slag, as Practised at the Port Pirie Works of the B.H.A.S.," by J. E. P. Murrie; "Determination of Fineness of Silver by Volhard's Method, as Practised at the Port Pirie Works of the B.H.A.S.," by T. A. Keats.

The Chairman reported that he had, as authorized, discussed the matter of reports in prospectuses with the Attorney-General for Victoria, who was now giving this difficult question consideration.

Resolved, as suggested by the Broken Hill Branch, that an effort be made to include Broken Hill in the itinerary of visits to be arranged in connection with the Pan-Pacific Congress, 1923.

Particulars of the areas which it is suggested should be included in the proposed geological survey of North-Western New South Wales were submitted, and authority given to approach the Department of Mines in the matter.

A report of progress in connection with the Port Pirie meeting was submitted.

Resolved that a list of disengaged members in each State be supplied to the Secretary of the Federal Engineering Council, and that duplicate lists be kept in each State.

Suggested alterations to the Chemical Engineering Diploma courses and entrance requirements of the Sydney Technical College were submitted, and referred to a Special Meeting for consideration.

Appointment of delegates to the General Council of the Australasian Association for the Advancement of Science was postponed.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holiays.

MEMBERSHIP.

Since the last issue of Proceedings the following have been admitted to the Institute :—

Associate Member :

TRUSCOTT, HENRY JAMES, c/o Block 10 Mine, Broken Hill, N.S.W.

Students :

CERUTTY, WILLIAM EDWARD, 415 Chapple-street, Broken Hill, N.S.W.
FORSTER, JOHN ARTHUR BLACKETT, Pressland House, Melville-street, Hobart, Tas.

PUBLICATIONS.

Most of the papers appearing in this issue were presented in proof form at the First Ordinary Meeting, in September. Written discussion on these papers is invited.

ELECTION OF PRESIDENT AND COUNCIL FOR 1923.

As notified by Circular No. 4, dated 11th October, 1922, Mr. Colin Fraser, of Melbourne, was the only nomination for the position of President ; he will, therefore, take office on 1st January, 1923. The following were elected to the Council unopposed :—*Victoria*—G. H. Broome and Professor E. W. Skeats ; *Queensland*—A. A. Boyd ; *Western Australia*—Rd. Hamilton ; *Tasmania*—R. M. Murray. Ballots for election of representatives of Australasia and New South Wales will be examined on 30th November, 1922. Those elected will take office on 1st January, 1923.

COMMONWEALTH ENGINEERING STANDARDS ASSOCIATION.

A recent issue of the *Commonwealth Government Gazette* contained the notification that the following have been appointed to represent the Australasian Institute of Mining and Metallurgy Incorporated on the main Committee of the Commonwealth Engineering Standards Association to work in co-operation with the Commonwealth Institute of Science and Industry, under the *Science and Industry Act* 1920 :—W. E. Wainwright, G. H. Blakemore, and F. Danvers Power. The first meeting of the main Committee will be held in Sydney on 2nd November, 1922.

REPORTS IN PROSPECTUSES.

The final recommendations of the Council on this subject are published under " Minutes of Meetings," 10th July, 1922. Steps have been taken to further these recommendations.

FIRST ORDINARY MEETING, 1923.

The Council has decided that the First Ordinary Meeting, 1923, be held in Tasmania. The precise locality of the meeting will be fixed later.

INTERNATIONAL MINING EXHIBITION.

The Sixth International Mining Exhibition will be held in the Royal Agricultural Hall, London, from 1st to 14th June, 1923. Previous exhibitions under the same management were held in London in 1903, 1904, 1906, 1908, and 1913. The Aust. I.M.M. has accorded its patronage to this exhibition, and earnestly desires that members present in London at the time should pay a visit to the exhibition. Full particulars may be obtained from Miss Edith A. Browne, Overseas Delegate, 43 Essex-street, Strand, W.C. 2.

AUSTRALASIAN ASSOCIATION FOR THE ADVANCEMENT OF SCIENCE.

The sixteenth meeting of the above Association will be held at the Victoria University College, Wellington, N.Z., commencing on 9th January, 1923. Lectures, entertainments, and excursions are being arranged. The membership fee is £1, which includes copy of report. Concessions in fares on boats and railways will be arranged. Further particulars may be obtained from the local secretaries of the A.A.A.S., or from the Secretary, Aust. I.M.M.

PROPOSED FEDERAL GEOLOGICAL SURVEY.

In August, 1921, the Australian National Research Council appointed a committee to consider the establishment of a Federal Geological Survey. The Aust. I.M.M. is represented on this committee. The Research Council recommended that a Federal Geological Survey be established. Draft recommendations, to which members of Council of the Aus. I.M.M. contributed, have been prepared and circulated, and are now under review.

GEOLOGICAL SURVEY OF NORTH-WESTERN NEW SOUTH WALES.

Recognizing the value of the excellent " Memoir on the Geology of Broken Hill," by Mr. E. C. Andrews, recently published by the

Department of Mines, N.S.W., the Council of the Aus. I.M.M. is now advocating the extension of this work. The valuable data resulting from surveys such as the above should greatly assist prospecting and the development of the mineral resources of the country.

TECHNICAL COLLEGE SYLLABUS.

Messrs. G. H. Blakemore, F. Danvers Power, Wm. Poole, and F. A. Eastaugh have been appointed to represent the Institute at a conference with representatives of the Sydney Technical College to discuss the diploma course in metallurgy.

MOVEMENTS OF MEMBERS.

B. V. BARTON has returned from Rhodesia, and is now in New Guinea reporting on an alluvial gold property.

C. F. COURTNEY, General Manager of the Sulphide Corporation, sailed for London early in November. He expects to return to Australia on a visit in a few years' time.

E. B. DOW, recently on the staff of the Block 10 Misima Gold Mines N.L., Misima, Papua, is leaving Papua on account of the closing down of the mine.

J. A. DUNN, who has been for some weeks in Melbourne on furlough, is returning to India to resume his work on the Indian Geological Survey.

F. R. HOCKEY has been transferred from Whyalla, S.A., to the Broken Hill Proprietary's Steel Works at Newcastle, N.S.W.

J. C. MOULDEN, of Seaton Carew, England, is now on a visit to Australia.

R. R. MARSHALL, of the Sulphide Corporation Ltd., has been transferred from the Central Mine, Broken Hill, to the Corporation's Melbourne office, 34 Queen-street.

P. R. OSBORNE, of the New Guinea Copper Mines Ltd., Tahira, Bootless Inlet, Papua, is at present in Melbourne.

A. E. KITSON (Member), C.B.E., F.G.S., Director of the Geological Survey, Gold Coast, West Africa, has been created C.M.G. by His Majesty the King. Mr. Kitson was appointed Senior Field Geologist and Assistant Director of Geological Survey of Victoria in 1905.

BRANCH REPORTS.

Broken Hill Branch.

An Ordinary Meeting was held on 4th July, 1922, at 8 p.m.

Mr. C. C. Freeman presided.

Minutes were confirmed and correspondence dealt with.

P. H. Warren read Part II. of paper prepared by himself and T. A. Read on "Methods and Apparatus for the Determination of Dust Suspended in Air." Discussion on the paper was contributed by T. G. Hanton, J. W. Salter, W. D. Whiteman, and C. H. J. Clayton.

A hearty vote of thanks was accorded Messrs. Warren and Read for their valuable paper.

A suggestion was made that an evening be reserved for discussion of paper already published in the Proceedings.

An Ordinary Meeting was held on 12th September, 1922, at 8 p.m.

Mr. F. Voss Smith presided.

Correspondence was received and minutes of previous meeting confirmed.

It was reported that a good attendance of Broken Hill members from Broken Hill was expected at the Port Pirie meeting.

Nomination of Councillors for the annual elections was considered.

A letter from Mr. F. Danvers Power *re* the Commonwealth Engineering Standards Association was read, and inquiries concerning the aims and objects of the Association directed to be made.

Details of the amended provisions for registration of architects were submitted.

Mr. C. C. Freeman read a paper entitled "Screening Tests on Fine Screens." Messrs. T. A. Read and F. G. White contributed to the discussion thereon.

Mr. T. B. Hanton read a paper entitled "Notes on Rock Drill Development on the Zinc Corporation Ltd." Mr. A. Fairweather contributed to the discussion thereon. It was resolved that to definitely crystallize Broken Hill ideas on the subject the secretary write to each mine on the field requesting their views.

Mr. C. H. J. Clayton read a paper entitled "The Chief Aim of the Federal Geological Survey." The author mentioned that prospecting of Australia's mineral resources should be the primary object of the survey. Mr. Voss Smith considered that general geology had equally good claims. Messrs. A. Fairweather and H. C. Harrison also contributed discussion.

Nomination of Mr. A. Lowry as a member was approved for transmission to the Council.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY
From 30th June, 1922 to 30th September, 1922.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	New York
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry: Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa: Journal	monthly	Johannesburg
Franklin Institute: Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers: Journal	monthly	London
Queensland Department of Mines: Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines: Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines: Report of Executive	monthly	Bulawayo
Queensland Department of Mines: Annual Report, 1921		Brisbane
Tasmanian Department of Mines: Annual Report, 1921		Hobart
New Zealand Board of Science and Art: Journal, Vol. V., Nos. 3 and 4		Wellington
Institution of Mining and Metallurgy: Bulletins, Nos. 213-215		London
Institution of Mining Engineers: Transactions, Vol. LXIII., Parts 3-5		London
Geological Society: Quarterly Journal, No. 310		London

Iron and Steel Institute :
    Journal, Vol. CV., Part 1, 1922 - - London
Institute of Chemistry of Great Britain and Ireland :
    Proceedings, Parts 1-4, 1922 - - London
Imperial Mineral Resources Bureau :
    Reports on Iron, Vanadium, Coal and Coke By-products; Mining Laws, South Africa; Laws and Regulations Relating to Lead Poisoning - London
R. Accademia delle Scienze dell' Institute di Bologna :
    Rendiconto, XXV. - - - - Bologna
Société des Ingeneiurs Civils de France :
    Bulletin, January, 1922 - - - Paris
Annales des Mines :
    Vol. I., 1922, Parts 5 and 6 ; Vol., II., 1922, Parts 7 and 8 - - - - - Paris
Geological Institution of the University of Upsala :
    Bulletin, Vol. XVIII., 1922 - - - Upsala
Mining and Geological Institute of India :
    Transactions, Vol. XVI., Part 3 ; XIII., Part 1 - Calcutta
Geological Survey of India :
    Records, Vol LIII., Part 3 ; LIV., Part 1 - Calcutta
Department of Mines, Canada :
    Reports on Barium and Strontium in Canada, 1922 Ottawa
Geological Survey of Alabama :
    Statistics of Mineral Production 1919 and 1920 ; Museum Bulletin, No. 5 - - - Alabama
California State Mining Bureau :
    State Mineralogist's Report, Vol. 18, Nos. 1-7 ; Californian Oil Fields, Vol. 7, Nos. 1-12 - San Fransisco
Secretaria de Industria Y Comercio :
    Boletin Dept. of Mines, Vol. XIII., Nos. 2-4 - Mexico
Institute Geologico de Mexico :
    Bulletin No. 37 - - - - Mexico
Cuerpo de Ingenieros de Minas :
    Bulletin, Nos. 99-103 - - - Lima

RECENT ARTICLES ON MINING MATTERS.

(30th June, 1922 to 30th September, 1922.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.
(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.
(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.
(4) *The Mining Journal*, London, E.C., wk., 6d.
(5) *Engineering and Mining Journal Press.*
(6) *Annales des Mines*, Paris, France, mth.
(7) *Publications*, Department of Mines, Melbourne, Victoria.
(8) *Publications*, Department of Mines, Sydney, New South Wales.
(9) *Publications*, Department of Mines, Adelaide, South Australia.
(10) *Publications*, Department of Mines, Brisbane, Queensland.
(11) *Publications*, Department of Mines, Perth, Western Australia.
(12) *Publications*, Department of Mines, Hobart, Tasmania.
(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.
(14) *Publications*, Bureau of Mines, Toronto, Ontario.
(15) *Publications*, Geological Survey of India, Calcutta.
(16) *Publications*, Geological Survey, U.S.A., Washington.
(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.
(18) *Publications*, California State Mining Bureau, Sacramento, Cal.
(19) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.
(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.
(21) *Quarterly Journal*, Geological Society, London.
(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.
(23) *Transactions*, Inst. Min. Eng., London.
(24) *Journal*, Canadian Mining Inst., Ottawa, Ontario.
(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.
(26) *Transactions*, Am. Inst. of Min. & Met. Eng., New York City.
(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.
(28) *Journal*, Franklin Inst., Philadelphia, Pa.
(29) *Chemical Engineering and Mining Review.*
(30) *Transactions*, Am. Soc. C.E., New York City.
(31) *Bulletins*, Société des Ingénieurs Civils, Paris.
(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.
(33) *Publications*, Iron and Steel Institute, London.
(34) *Proceedings*, Inst. of Mech. Eng., London.
(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.
(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.
(37) *Transactions*, Mining and Geological Institute of India, Calcutta.
38) *Publications*, Department of Mines, Wellington, N.Z.
(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

LIST OF ARTICLES.

MECHANICAL.

Friction Factors in Ventilating Pipe. (5) June 10, 1922.
Seneca's Novel Mine Cage. H. M. Payne. (5) June 17, 1922.
A Wilfley Centrifugal Sand Pump. (5) July 1, 1922.
A Problem in Mine Ventilation. C. N. Schuette. (5) July 8, 1922.
A Continuous Weighing Machine. (5) July 8, 1922.
Pulverized Coal at the Bunker Hill and Sullivan Smelter. H. M. Payne. (5) July 22, 1922.
The Hawkesworth Detachable Drill Bit. F. E. Wormser. (5) Aug. 5, 1922.
Testing Rock Drills. R. J. Day. (5) Aug. 19, 1922.
A Mechanically-Operated Air Door. T. A. Miller. (5) Aug. 26, 1922.
Chain and Bucket Elevator Device. A. W. Allen. (5) Sep. 23, 1922.

METALLURGICAL.

The Ajo Enterprise of the New Cornelia Copper Co. A. W. Allen. (5) Serial commenced June 3, 1922.
Assay of By-Products of Gold Reduction Works. J. Watson. (5) July 29, 1922.
Treating Surface Residues for Gold. (5) July 29, 1922.
Mechanical Sampling of Ore. C. D. Demond and A. C. Halferdahl. (5) Aug. 12, 1922.
Sulphatizing Roasting of Copper Ores. (5) June 10, 1922.
Non-Cupellation Method of Gold Bullion Assay. A. W. Allen. (5) June 17, 1922.
Sulphur-Dioxide Leaching of Copper Ores. (5) June 24, 1922.
Copper Content and Specific Gravity of Copper Mattes. (5) June 24, 1922.
Development of Metallurgical Practice at the Hollinger Mine. P. A. Robbins. (5) June 24, 1922.
A Proposed Plant for the Electrothermic Dry Distillation of Zinc Ores. C. H. Fulton. (5) July 1, 1922.
Surface-Tension Phenomena and Electrostatics. R. H. James and D. W. Leeke.
Distributing Granular Material in Drainage Vats. A. W. Allen. (5) July 1, 1922.
Air-Separating Mill which Produces a 200-Mesh Product. (5) July 1, 1922.
Electrothermic Dry Distillation of Zinc Ores. W. McA. Johnson. (5) Aug. 5, 1922.
Ammonia Leaching of Zinc Ores. J. F. Cullen and T. E. Harper. (5) Aug. 5, 1922.
Milling Graphite at Stockertown, Pa. A. B. Parsons. (5) Aug. 19, 1922.
The Wood Gravity Classifier. (5) Aug. 19, 1922.
Leaching Ore with Sulphur Dioxide. J. Irving. (5) Aug. 26, 1922.
A Resume of Electrolytic Zinc Practice. J. T. Ellsworth. (5) Sept. 2, 1922.
An Illustrated History of Mining Metallurgy. H. H. Manchester. (5) Serial commenced Sept. 2, 1922.
Ferric Salts as Solvents in the Leaching of Roasted Copper Ores. P. R. Middleton. (5) Sept. 9, 1922.
Sintered Concentrates Now Smelted in Mount Lyell Blast Furnaces. (5) Sept. 9, 1922.
The Development of the Vertical Disk Crusher. D. Cole. (5) Sept. 23, 1922.
Aims of the A.I.M.E.'s Committee on Milling Methods. E. A. Hersam. (5) Sept. 16, 1922.
Determining Relative Tonnages from Assays. A. T. Fray. (5) Sept. 16, 1922.
Gold Precipitation by Zinc Dust. A. Newton and L. L. Fewster. (25) June, 1922.
The Application of Flotation to the Antimonial Gold Ores of the Murchison Range. H. R. Adam. (25) July, 1922.
Investigations on the Copper-Nickel Ores of the Rustenburg District. J. A. Ortlepp. (25) Aug., 1922.
Copper Leaching at the Whim Well, Western Australia. H. R. Sleeman. (32) July, 1922.
The Nevill-Soanes Copper Process. P. W. Nevill. (32) Aug., 1922.
Leaching with Sulphur Dioxide. C. E. Van Barneveld and E. S. Leaver. (32) Sept., 1922.
Basic Principles of Gravity Concentration. T. Simons. (26) Bull., July, 1922.
Factors Controlling Capacity of Rock Crushers. E. A. Hersam. (26) July, 1922.

MINING.

Using Residue as Stope Filling. A. W. Allen. (5) July 15, 1922.
Grouting Water Channels Underground. T. A. Rickard. (5) Aug. 12, 1922.
Promoting Efficiency in Drilling and Blasting. B. F. Tillson. (5) Sept. 2, 1922.
Mining Barytes in Ireland. F. H. Ahier. (5) Sept. 2, 1922.
Driving Drifts and Small Crosscuts. D. M. Houston. (5) Sept. 9, 1922.
Shot-Firing by Electricity. N. S. Greensfelder. (26) Bull., Sept., 1922.

MISCELLANEOUS

The Case For and Against a Metal Exchange. F. E. Wormser. (5) Sept. 9, 1922.
Driving an Eighteen-Mile Tunnel Through the Sierra Nevada. A. B. Parsons. (5) July 15, 1922.
Marketing of Barytes. S. W. Thompson. (5) July 15, 1922.
Marketing of Magnesite. R. W. Stone. (5) July 29, 1922.
Geography and Burma Mines Limited. A. B. Parsons. (5) July 29, 1922.
The Dust Problem in Dry Crushing. (5) Aug. 12, 1922.
Marketing of Asbestos. B. Marcuse. (5) Aug. 12, 1922.
Oil Prospects in Western Australia. (5) Aug. 12, 1922.
Two and Three-Shift Day. (5) Aug. 12, 1922.
How Should Depletion and Depreciation be Charged? T. O. M'Grath (5) June 10, 1922.
The Problem of Depletion. H. B. Fernald. (5) June 17, 1922.
The Bawdwin Mine in Burma. A. B. Calhoun. (5) June 24, 1922.
Marketing of Sulphur. A. G. Wolf. (5) July 1, 1922.
Mining Curriculum of the Massachusetts Institute of Technology. G. J. Young. (5) July 8, 1922.
Some Problems in Mine Accounting. G. E. H. Goodner. (5) July 22, 1922.
Diamond Drilling for Petroleum. E. R. Lilley. (5) July 22, 1922.
Mine Labour. H. J. C. M'Donald. (5) Aug. 5, 1922.
Surface Tension and Flotation Phenomena. D. W. Leeke and R. H. Jarvis. (5) Aug. 19, 1922.
Drill Sampling of Dredge Ground. R. T. Hancock. (5) Aug. 19, 1922.
Use of Lead is Increasing. F. E. Wormser. (5) Aug. 19, 1922.
The Use of Telephones in Mines. D. E. A. Charlton. (5) Aug. 26, 1922.
The Marketing of Manganese. W. R. Crane. (5) Aug. 26, 1922.
Cutting Mine Samples with Machines. D. C. Gilbert. (5) Aug. 26, 1922.
Should Depletion and Depreciation be Charged? R. M. Hughes. (5) Sept. 9, 1922.
The Price of Radium in Relation to the Cost of Mining. G. Kunkle. (5) Sept. 16, 1922.
New Uses for Selenium and Tellurium. (5) Sept. 16, 1922.
Miners' Hookworm at the Centenillo Mines, Spain. G. S. Martin. (22) Bull., July, 1922.
Statistics of Extremely Depreciated Currency. R. A. Lehfeldt. (25) Aug., 1922.
Notes on Organization for Promoting Industrial Efficiency. S. de Smidt. (25) Aug., 1922.
Iron Ore Resources of the British Empire. H. Louis. (32) Sept., 1922.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED)

NEW SERIES.

No. 48.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

31st DECEMBER, 1922.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

LIST OF OFFICERS, 1922.

President:

GEORGE C. KLUG, W.A.

Vice-Presidents:

COLIN FRASER, S.A. | D. B. WATERS, N.Z.

Councillors:

Representing Australasia—

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- G. D. DELPRAT
- COLIN FRASER
- JAMES HEBBARD
- JAMES HORSBURGH
- A. S. KENYON
- JOHN W. MOULE
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- GEO. H. BROOME
- H. HERMAN
- E. W. SKEATS

Representing Western Australia—

- RICHARD HAMILTON
- J. W. SUTHERLAND

Representiug Tasmania—

- H. W. GEPP

Ex officio Member of Council—

R. W. CHAPMAN, S.A. (Past President.)

Hon. Treasurer:

A. S. KENYON.

Bankers:

COMMONWEALTH BANK OF AUSTRALIA, Collins Street, Melbourne.

Secretary:

D. L. STIRLING, Collins Court, 374 Little Collins Street, Melbourne, Vic.

BRANCH SECRETARIES:

Broken Hill -	C. H. J. CLAYTON, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie -	S. G. TURRELL, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan -	B. G. PATTERSON, Mount Morgan, Q.
Sydney -	F. DANVERS POWER, 31 Hunter Street, Sydney, N.S.W

LOCAL CORRESPONDENTS:

South Australia -	H. W. GARTRELL, The University, Adelaide.
New Zealand -	D. B. WATERS, The University, Dunedin.
Tasmania -	RUSSELL M. MURRAY, Queenstown.
North Queensland	MURRAY RUSSELL, Department of Mines, Brisbane.
Papua - -	W. G. LANGFORD, Oilfields, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

COLLINS COURT, 374 LITTLE COLLINS STREET, MELBOURNE, VICTORIA.

Institute Matters.

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MINUTES OF MEETINGS

OF THE EXECUTIVE COMMITTEE

Summary.

AUGUST 28TH, 1922.—1 P.M. (Special Meeting).

Suggested alterations to the Chemical Engineering Diploma courses and entrance requirements of the Sydney Technical College were discussed, and it was resolved—

> " That the Executive Committee of the Australasian Institute of Mining and Metallurgy is of the opinion that, while a knowledge of French and German for technical college students is desirable, in the present state of development of technical education in Australia it would be premature to make a knowledge of these languages a pre-requisite to the issue of a diploma in chemistry or chemical engineering, and, if insisted upon, would prejudicially affect the careers of numerous students who would otherwise be able to play a useful part in technical industry. That the Executive Committee is of the opinion that facilities for the study of French and German should be given in technical colleges, and that the passing of

these subjects either at the technical colleges or at examinations of suitable standards might excuse candidates for the diploma from certain subjects of the course which would be regarded for this purpose as equivalent to them."

SEPTEMBER 13TH, 1922.—1 P.M. (Special Meeting).

The secretary reported that the following nominations of office-bearers had been received in response to Circular No. 2 :—President: Colin Fraser. Council : E. G. Banks, John M. Bridge, G. D. Delprat, James Horsburgh, and John W. Moule (Australasia) ; G. H. Blakemore, H. A. Evans, Wm. Poole, and F. Voss Smith (N.S.W.) ; G. H. Broome and Professor Skeats (Vic.) ; A. A. Boyd (Q.) ; Rd. Hamilton (W.A.) ; R. M. Murray (Tas.)

Resolved that the date of ballot be 30th November, 1922, and that only New South Wales members vote for New South Wales representatives.

OCTOBER 9TH, 1922—1 P.M.

The Secretary's report was presented, and accounts to the amount of £120, including postage, £9, salaries and rent (two months), £50, blocks for Proceedings, £15, commission, £4. Expenses Port Pirie meting (£35), were passed for payment.

The following were nominated delegates to the Council of the Australasian Association for the Advancement of Science, Wellington meeting :—Professor Skeats, Professor Waters, Messrs. E. G. Banks, A. S. Kenyon and G. D. Delprat.

The Secretary's report of the First Ordinary Meeting, held in September, was submitted and received and a vote of thanks recorded to the Broken Hill Associated Smelters Ltd. and the Broken Hill Proprietary Ltd. for their valuable co-operation.

The following recommendations to the Council from the First Ordinary Meeting were submitted :—

(1) That indexing mining literature be again urged by the Council.

(2) That the Institute should be represented on all commissions, etc., dealing with the mining industry or any phase thereof.

(3) That the Council should do its utmost to further the proposed Federal geological survey as an aid to prospecting, and also consider whether the Federal geological survey should specialize along oil-prospecting lines.

(4) That the Federal Engineering Council be requested to get into touch with the various authorities and urge that concessions in fares be granted to members of the Institute travelling from the various States to meetings, even if less than six (6) travel from any particular State.

(5) That the locality of the First Ordinary Meeting, 1923, be left to the Council to decide.

Resolved that items (1), (2), and (3) be deferred. Item No. 4 was approved. With regard to item No. 5, it was resolved that the First Ordinary Meeting, 1923, be held in Tasmania.

A report of meeting of the Broken Hill Branch was submitted and received.

Resolved that copies of proposal *re* geological survey of North-Western New South Wales be forwarded to members of Council for expression of opinion thereon.

Messrs. F. Danvers Power, G. H. Blakemore, Wm. Poole, and F. A. Easthaugh were nominated to represent the Institute on the Advisory Committee of Sydney Technical College.

A Paper by Messrs. C. C. Freeman and H. J. Hookings on "Screening Tests on Fine Screens" was referred to the Publication Committee.

Resolved that the Institute grant its patronage, as invited, to the International Mining Exhibition, to be held in London in June, 1923.

Applications for admission to the Institute were submitted and dealt with as follows :—As Members.—A. H. P. Lowry (approved for ballot); F. W. Reid (approved for ballot). As Junior.—O. J. Carlson (deferred).

Mr. W. E. Wainwright reported progress in regard to the Engineering Standards Association.

NOVEMBER 20TH, 1922.—1 P.M.

The secretary's report was presented, and accounts to the amount of £306, including printing (£178), furnishing (£43), process blocks

(£22), salary and rent, two months (£50), and petty expenses (£11) were passed for payment.

The Chairman declared the following elected to the Institute by ballot of members of Council :—As Members—Edward William Finch and Percival Faraday Thompson. As Associate Member—James Bain Auld. The following students were admitted :—Dick Beal and Alfred Lawrence Eriksen.

Papers as follow were referred to the Publication Committee :—"Some Calculations Dealing with Mine Timber," by H. F. Pearson; "Ventilating Metalliferous Mines : a Method of Attack," by Thos. G. Hanton.

A report of a meeting of the Broken Hill Branch was submitted and received.

Scrutineers were appointed to examine ballots for election of Councillors on 30th November, 1922.

It was reported that the main committee of the Engineering Standards Association had been appointed by the Government, and the appointment gazetted on 12th October, 1922. Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power represent the Aus. I.M.M. on the main committee, the first meeting of which concluded on 3rd November, 1922.

Resolved that arrangements be now completed for interviewing the Minister for Mines, New South Wales, with the object of urging the extension of the geological survey of Broken Hill.

Further suggestions from members of Council in regard to the proposed Federal Geological Survey were submitted, and the secretary was instructed to prepare a draft memorandum on the subject, to be submitted, when revised, to the committee appointed by the Australian National Research Council to deal with the proposed Federal Geological Survey.

Other routine business was transacted.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holiays.

MEMBERSHIP.

Since the last issue of Proceedings the following have been admitted to the Institute :—

Members.

Names and Addresses.

FINCH, EDWARD WILLIAM, 12 Loftus-street, Sydney, N.S.W.
THOMPSON, PERCIVAL FARADAY, The University, Melbourne, Vic.

Associate Member.

AULD, JAMES BAIN, Crusington, N.Z.

Students.

BEAL, DICK, British Mine Residence, Broken Hill, N.S.W.
ERIKSEN, ALFRED LAWRENCE, 336 Thomas-street, Broken Hill, N.S.W.

ANNUAL MEETING, 1923.

The Annual Meeting, 1923, will be held at the Institute Rooms, 374 Little Collins-street, Melbourne, on Monday, 26th March, at 1 p.m. The business of the meeting will include (1) consideration and adoption of the Annual Report of the Council and Statement of Accounts for 1923 ; (2) appointment of Auditor for 1923.

ELECTION OF PRESIDENT AND COUNCIL, 1923.

As a result of ballot held on 30th November last the undermentioned were elected to fill the annual vacancies :—

President : Colin Fraser.

Council—Australasia : E. G. Banks, John M. Bridge, G. D. Delprat, and Jas. Horsburgh.
New South Wales : G. H. Blakemore, H. A. Evans, and F. Voss Smith.
Victoria : G. H. Broome and Professor E. W. Skeats.
Queensland : A. A. Boyd.
Western Australia : R. Hamilton.
Tasmania : R. M. Murray.

ENGINEERING CONFERENCE, ADELAIDE.

Members of the Australasian Institute of Mining and Metallurgy are cordially invited to attend a meeting of the Institution of

Engineers, Australia, to be held in Adelaide from 6th to 12th March inclusive. An interesting programme of visits, lectures, etc., is being arranged. Members wishing to attend should advise Mr. E. F. Eberbach, Town Hall, Adelaide, who will supply full particulars regarding railway concessions, hotel accommodation, etc.

PROPOSED FEDERAL GEOLOGICAL SURVEY.

Further suggestions from members of Council in regard to the proposed Federal Geological Survey have been received, and a memorandum is being prepared on the subject for reference to the Australian National Research Council.

PAN-PACIFIC CONGRESS.

The Institute has been officially notified that the next Pan-Pacific Congress will be held in Australia in August, 1923. The meeting will be opened in Melbourne on 13th August and terminate there on 22nd August, when members will leave for Sydney, where the meeting will extend from 23rd August to 3rd September. The president of the Congress is Professor D. Orme Masson ; Mr. R. H. Cambage is hon. secretary-treasurer, and Associate Professor A. C. D. Rivett joint hon. secretary. The Australasian Institute of Mining and Metallurgy has received an invitation to be represented at the Congress. The Council of the Aus. I.M.M. has already suggested that a visit to Broken Hill be included in the itinerary.

COMMONWEALTH ENGINEERING STANDARDS ASSOCIATION.

Professor W. H. Warren (Dean of the Faculty of Engineering at the Sydney University) has been elected honorary chief technical adviser to the above association, and Mr. N. J. Lacey honorary technical secretary. Important developments in connection with the matter of standardization during 1923 are expected.

DEVICE FOR RECORDING SHAFT SIGNALS.

The Rand Mutual Assurance Co., Johannesburg, Transvaal, offers prize-money not exceeding £250 to anyone for design or designs of the most practicable device for recording shaft signals, etc., on the mines. The device must be capable of making a permanent record showing clearly and separately (1) the time and nature of all signals received by the driver—(*a*) from underground, (*b*) from the bank; (2) the time and nature of all signals given by the driver (*a*) to underground, (*b*) to the bank; (3) the conveyance or compartment (if any) to which the signals mentioned in (1) and (2) refer; (4) the time during which each conveyance is in motion. Conditions will be supplied on application to the secretary, Aus. I.M.M., Melbourne.

MOVEMENTS OF MEMBERS.

A. S. ARMSTRONG has left the Rhodesian Lead and Zinc Syndicate, Broken Hill, Rhodesia, to take over the management of the Chillagoe Smelters, North Queensland.

E. G. BOYD is now on the staff of the North Broken Hill mine.

H. EDWARD CLAYTON is in Papua, having accepted the position of mine superintendent of the Laloki mine.

EDMUND B. DOW, recently with the Block 10 Co., Misima, has rejoined the staff of the Sulphide Corporation Ltd. as surveyor at the Central Mine, Broken Hill.

COLIN FRASER (President, 1923) will leave on a visit to London in March next.

H. LIPSON HANCOCK has resigned his position as general manager of the Wallaroo and Moonta Mining and Smelting Co., and is now residing at Gawler Park, Angaston, S.A.

A. L. HAY, general manager of the Block 10 Misima Co., is in Melbourne, his company having ceased operations.

VINCENT N. JONES is now at Beaudesert, Queensland.

G. C. KLUG (President, 1922), who has been visiting Europe and America, will return to Melbourne in January, 1923. Whilst in London he conferred with the Institution of Mining and Metallurgy on matters affecting the welfare of both Institutes.

E. H. B. MACARTNEY, B.C.E. (Melb.), is now residing at 36 Bagot-road, Subiaco, W.A.

W. H. MAWDSLEY'S present address is 36 Barnard-street, North Adelaide. He was for many years on the staff of the Mount Morgan Gold Mining Co.

W. J. NEWBIGIN has left Melbourne, and is now at 175 Clarence-street, Sydney.

A. Percy Penman, B.E. (Sydney), is practising as mining, metallurgical, and civil engineer at Intercolonial House, Castlereagh-street, Sydney.

Harold Rabling, B.M.E. (Melb.), after spending some years in U.S.A. on the lead and zinc mines in Missouri, has returned to Melbourne. He was mill superintendent of the St. Joseph Lead Co., Bonne Terre, Missouri.

Hector Stewart, M.C.E. (Melb.), of Perth and Wagin, W.A., will be visiting Melbourne in January.

E. A. White, manager of the Electrolytic Refining and Smelting Co. at Port Kembla, N.S.W., will shortly leave for America on a business trip on behalf of the company.

BRANCH REPORTS.

BROKEN HILL BRANCH.

An ordinary meeting was held on 17th October, 1922, at 8 p.m.

Mr. C. C. Freeman presided.

Correspondence on the subject of the scope and organization of the Commonwealth Engineering Standards Association was received.

A circular letter *re* rock drills, issued by the secretary to each mine, was read, and, as only one reply had been received to date, the secretary urged that each mine send along its views so that some standard (representative of Broken Hill practice) could be fixed.

Extracts of interest from minutes of Council meetings were read, and special interest shown in regard to a letter from Mr. E. C. Andrews, Government Geologist, N.S.W., on the subject of the extension of the geological survey of Broken Hill.

The secretary (Mr. Clayton) was requested to make inquiries from the head office *re* the administration of mining bursaries.

The secretary expressed regret that, owing to the absence of members of Council at the Port Pirie meeting, questions of policy and scope of the Institute for the ensuing year were not discussed.

A recommendation was made that no students below the minimum age limit fixed for student members of the Institute be allowed to attend First Ordinary Meetings.

A paper by Mr. Thos. G. Hanton dealing with "The Ventilation of Metalliferous Mines," Part II., was read and discussed.

The chairman announced that reply to discussion would be submitted at a later meeting.

An ordinary meeting was held on Tuesday, 7th November, 1922, at 8 p.m.

Mr. C. C. Freeman presided.

Notes on the Commonwealth Engineering Standards Association were submitted.

Further replies *re* rock-drill practice were submitted, and action deferred.

Mr. Fairweather explained in detail the operation of the Mining Bursaries Committee. The opinion was expressed that this was a matter which should be administered by the Institute in order that scholars might get the best type and variation of experience.

Mr. Hanton read his reply to discussion on his paper, "Ventilation of Metalliferous Mines."

A paper was read by Mr. H. F. Pearson, entitled "Some Calculations Dealing with Mine Timber."

A discussion then took place on Mr. F. J. Mars' paper, "Electrical Power Application on the Broken Hill Mines, with Particular Reference to Remote Motor Control and Electrical Hoisting at the South Mine."

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 30th September, 1922 to 31st December, 1922.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	bi-monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
Royal Society of Victoria : Proceedings, Vol. XXXV., Part I.		Melbourne
Australian National Research Council : Australian Science Abstracts, Vol. I., Nos. 1-4		Sydney
Queensland Department of Mines : Geological Survey Publication, No. 270		Brisbane
Institution of Mining and Metallurgy : Bulletins, Nos. 216-219		London
Institution of Mining Engineers : Transactions, Vols. LXIII., No. 6 ; LXIV, Parts 1 and 2		London
Geological Society : Quarterly Journal, No. 311		London

Geologists' Association :
Proceedings, Vol. XXXIII., Parts 3 and 4 - London
Imperial Mineral Resources Bureau:
Reports on Coal and Coke By-products, 1913-1919; Iron Ore in Europe and Africa (Part 6), Nickel London
Department of Scientific and Industrial Research :
Report of the Tin and Tungsten Research Board - London
Royal Dublin Society :
Economic Proceedings, Vol. XI., No. 15 ; Scientific Proceedings, Vol. XVI., Nos. 35-39; Vol. XVII., Nos. 1-10 - - - Dublin
Societa Geologica Italiana :
Bulletin, XLI., Parts 1 and 2 - - Rome
Société des Ingeneiurs Civils de France :
Bulletins, April to September, 1922 - - Paris
Annales des Mines :
Vol. II., 1922, Parts 9-11 - - - Paris
Sveriges Geologiska Undersökning :
Sveriges Olenidskiffer by A. H. Westgard - Stockholm
Geological Survey of India :
Records, Vol. LIV., Part 2 - - - Calcutta
Department of Mines and Geology, Mysore :
Report of Chief Inspector of Mines, 1920-21 - Mysore
Department of Mines, Canada :
Summary Report, 1921, Parts A and D ; Report on Structural Materials along St. Lawrence River ; Talc and Soapstone in Canada - Ottawa
United States Geological Survey :
Professional Papers—Nos. 121, 123, 128 E, 129 A to I
Bulletins—Nos. 679, 704, 713, 718 KLM, 716 GH, 719, 721, 725 F to J, 726, 726 A to G, 730 AB, 735 ABC, 736 A
Mineral Resources—1918, Part I, No. 3, Part II, Nos. 31, 35, 37 ; 1919, Part I, Nos. 9-13, 15, 17, 20-29, Part II, Nos. 13-19, 21, 28-35 ; 1920, Part I, Nos. 3-29, Part II, Nos. 2, 4-31 ; 1921, Part I, Nos. 1-5, Part II, Nos. 1-5 - New York

American Institute of Mining and Metallurgical Engineers :	
“Mining and Metallurgy” (monthly). Various papers	New York
California State Mining Bureau :	
“Mining in California” (monthly) “Summary of operations California Oil Wells (monthly)	San Francisco
Secretaria de Industria Y Comercio :	
Bulletin Vol. XIII., No. 5	Mexico

RECENT ARTICLES ON MINING MATTERS.

(30th September,, 1922 to 31st December, 1922.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.
(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.
(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.
(4) *The Mining Journal*, London, E.C., wk., 6d.
(5) *Engineering and Mining Journal Press.*
(6) *Annales des Mines*, Paris, France, mth.
(7) *Publications*, Department of Mines, Melbourne, Victoria.
(8) *Publications*, Department of Mines, Sydney, New South Wales.
(9) *Publications*, Department of Mines, Adelaide, South Australia.
(10) *Publications*, Department of Mines, Brisbane, Queensland.
(11) *Publications*, Department of Mines, Perth, Western Australia.
(12) *Publications*, Department of Mines, Hobart, Tasmania.
(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.
(14) *Publications*, Bureau of Mines, Toronto, Ontario.
(15) *Publications*, Geological Survey of India, Calcutta.
(16) *Publications*, Geological Survey, U.S.A., Washington.
(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.
(18) *Publications*, California State Mining Bureau, Sacramento, Cal.
(19) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.
(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.
(21) *Quarterly Journal*, Geological Society, London.
(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.
(23) *Transactions*, Inst. Min. Eng., London.
(24) *Journal*, Canadian Mining Inst., Ottawa, Ontario.
(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.
(26) *Transactions*, Am. Inst. of Min. & Met. Eng., New York City.
(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.
(28) *Journal*, Franklin Inst., Philadelphia, Pa.
(29) *Chemical Engineering and Mining Review.*
(30) *Transactions*, Am. Soc. C.E., New York City.
(31) *Bulletins*, Société des Ingénieurs Civils, Paris.
(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.
(33) *Publications*, Iron and Steel Institute, London.
(34) *Proceedings*, Inst. of Mech. Eng., London.
(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.
(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.
(37) *Transactions*, Mining and Geological Institute of India, Calcutta.
(38) *Publications*, Department of Mines, Wellington, N.Z.
(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

LIST OF ARTICLES.

MECHANICAL.

Counterbalancing a Small Hoist at the Hibernia Mine. F. E. Wormser. (5) Oct. 21, 1922.

Pipe Friction and Pump Efficiency. W. Brazenall. (23) Vol. LXIII., Part 1.

Experiments with Single-Inlet, Duplicate, and Double-Inlet Cupell Fans. E. S. Wood. (23) Vol. LXIII., Part I.

Air-Cooling Plant at the Morro Velho Mine of the St. John Del Rey Mining Co. Ltd., Brazil. E. Davis. (23) Vol. LXIII., Part 5.

Silver Mining and Milling at Talache, Idaho. F. E. Wormser. (5) Sept. 30, 1922.

METALLURGICAL.

The Sundt-Diaz Flotation Machine. F. A. Sundt. (5) Sept. 30, 1922.

Pyritic Smelting at the Siemens–Kwarzchana Copper Works, in the Caucasus. C. Offerhaus.. (5) Sept. 30, 1922.

Sodium Sulphide as a Differential Flotation Reagent. J. B. Parker. (5) Oct. 7, 1922.

Proposed Pyritic Smelting in a Reverberatory. E. C. King. (5) Oct. 7, 1922.

Distributing Box for Dorr Thickener. C. G. Dresser. (5) Oct. 7, 1922.

Ferric Salts as Solvents in the Leaching of Roasted Copper Ores. L. E. Roberts, G. S. Tilley, and O. C. Ralston. (5) Oct. 21, 1922.

The Greenawalt Electrolytic Copper Extraction Process. W. E. Greenawalt. (5) Oct. 21, 1922.

Gold Mining Developments in the Black Hills. F. E. Wormser. (5) Oct. 28, 1922.

Some Applications of Electrical Equipment in the Metallurgy of Nickel. J. L. M'K. Yardley. (5) Nov. 4, 1922.

The New Treatment Plant of the United Comstock Mines. G. J. Young. (5) Nov. 11, 1922.

Oxidation of Ferrous Salts by Atomized Air in Acid Solution Thought Practicable. W. E. Greenawalt. (5) Nov. 25, 1922.

Wet and Dry Grinding in Conical Mills. H. Hardinge. (5) Nov. 25, 1922.

Sampling Spotty Gold Ores. C. D. Demond and A. C. Halferdahl. (5) Nov. 25, 1922.

Surface Tension and Flotation Phenomena. A. H. Fahrenwald. (5) Dec. 9, 1922.

An Investigation of Losses in Copper Blast Furnace Slag. D. F. Stedman. (5) Serial commenced Dec. 9, 1922.

Objections to King Method of Pyritic Smelting in a Reverberatory. W. M'A. Johnson. (5) Dec. 16, 1922.

Dust in Milling Plants. M. W. von Bernewitz. (5) Dec. 16, 1922.

A Wood-Burning Furnace for Smelting Cyanide Precipitate. M. B. Evans. (5) Dec. 16, 1922.

Improved Flotation Machine. W. I. Nelson. (5) Dec. 16, 1922.

Electrical Precipitation in the Chemical Industry. P. E. Landolt. (39) Nov., 1922.

Basic Principles of Gravity Concentration: Discussion. (26) Dec., 1922.

Soluble Silica in the Preparation of Zinc-Sulphate Solution for Electrolysis. J. O. Betterton. (26) Dec., 1922.

Colloid Chemistry and Metallurgy. W. D. Bancroft. (26) Dec., 1922.

Use of Magnetic Ore in the Blast Furnace. G. B. Pilling. (26) Dec., 1922.

Ball and Roller Bearings, with Particular Reference to Mining Work. R. Allen. (23) Vol. LXIV., Part 2.

MINING.

Reversing Ventilating Current During Mine Fires or Explosions. M. W. von Bernewitz. (5) Nov. 18, 1922.

Diamond Drilling in El Potosi Mine. H. A. Walker. (5) Nov. 18, 1922.

Practical Manufacture and Use of Liquid Oxygen Explosive for Breaking Ore. A. D. Akin. (5) Dec. 2, 1922.

Mining Methods in the Mineville (N.Y.) District. E. C. Henry. (26) Oct., 1922.

Methods of Working the Barnsley Seam of the South Yorkshire Coalfield. H. Rhodes and M. Rhodes. (23) Vol. LXIII., Part 6.

MISCELLANEOUS.

Experiments in Electrical Prospecting. S. F. Kelly. (5) Serial commenced Oct. 7, 1922.

The Marketing of Tin. J. H. Lang. (5) Oct. 7, 1922.

The Tellurides. T. A. Rickard. (5) Serial commenced Oct. 21, 1922.

The Marketing of Platinum. J. M. Hill. (5) Oct. 21, 1922.

The Boarding Houses at Small Mines. C. N. Schuette. (5) Oct. 21, 1922.

Rand Ore Reserves. A. C. Key. (5) Oct. 21, 1922.

Bauxite · Occurrence, Mining, and Uses. R. B. Ladoo. (5) Nov. 4, 1922.

The Relation of Gold to Currency. H. R. Sleeman. (5) Nov. 18, 1922.

Metal Mine Fires. D. Harrington, B. O. Pickard, and H. M. Wolfin. (5) Nov. 25, 1922.

The Marketing of Quicksilver. H. W. Gould. (5) Dec. 9, 1922.

Some Notes on the Evolution of the Konimeter. R. A. H. Flugge and De Smidt.

Interpretation of Results of Coal-Washing Tests. T. Fraser and H. F. Yancey. (26) Nov., 1922.

Metal Mine Ventilation in the South-West. C. A. Mitke. (26) Oct., 1922.

The Aztec Mine, Baldy, New Mexico. C. A. Chase and D. Muir. (26) Oct., 1922.

The Rate of Absorption of Poisonous Amounts of Carbon Monoxide by the Blood. A. P. Veale. (23) Vol. LXIII., Part 6.

Rock Temperatures in the Coal Measures of Great Britain. J. I. Graham. (23) Vol. LXIII., Part 5.

Papers and Discussions.

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PAPERS—

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Gold Occurrences at Bendigo (with 1 Plate and 7 Figs.) By F. L. Stillwell, D.Sc. - - - - - - - - - - 1

GOLD OCCURRENCES AT BENDIGO.

BY F. L. STILLWELL, D.Sc.

INTRODUCTION.

IN previous discussions on the occurrence of gold in the Bendigo quartz reefs * the opinion has been put forward that the gold shoots are primary concentrations in the quartz reefs during the period of vein-formation. The view has been based upon the constant features of the gold shoots in a large number of individual reefs, independent of their depth below the surface. These features include the mineral associations accompanying the gold, the appearance and association of the gold in the quartz, the similarity in all respects (excepting mineral content) between richly gold-bearing quartz and poor quartz, and the variations in mineral content in the reef apart from gold. The view has been supported by a relation of the gold shoots to faults, together with the dependence of the reefs on the existence of pre-mineral faulting. The secondary deposition of gold has only been observed in a few instances associated with the younger monchiquite dykes, and is of no economic importance.

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* F. L. Stillwell. "The Factors Influencing Gold Deposition in the Bendigo Goldfield." *Commonwealth Advisory Council of Science and Industry. Bulls.*, Nos. 4, 8, 16.

The quartz reefs represent gradual growths by crystallization from circulating mineral solutions, accompanied by more or less replacement of the country rock. The gradual entry of the solutions into the mineral veins has taken place through porous beds, fractures, and faults, and the gold shoots are considered to be formed by the precipitation of the gold around the points of entry of the solutions into the reefs. Of the precipitating matter, the most important has been the carbonaceous residues of replaced slate.

The main gold shoots are thus considered to be primary concentrations within the main zones of deposition from mineral solutions which have circulated through an anastomosing plexus of faults, and their eccentricities of occurrence are attributed in part to the original eccentricities of circulation of the solutions. These views have been upheld in a subsequent experience of the Bendigo reefs, and more particularly of the Constellation reef, which takes rank among the richest of Bendigo's saddle reefs.

In the preparation of this paper free use has been made of the plans and records of the Bendigo Amalgamated Goldfields Company Ltd., and thanks are due to the management for making these available and for kindly interest throughout.

CONSTELLATION REEF.

The Constellation reef occurred along the northern portion of the Garden Gully anticline, close to the town of Eaglehawk. It was discovered in September 1918, and since that date the gold shoot has been practically exhausted, producing 60,649 ozs. for 43,037 tons over a length of 900 ft. The gold was first struck in the cap as the result of the extension to centre country of the west crosscut at the 723-ft. level, from the Constellation shaft. The reef had previously been intersected 300 ft. further south by the main crosscut from the Golden Pyke shaft at the 830-ft. level, where it had been tested by 460 ft. of driving, rising and winzing. This work was not rewarded by the discovery of payable gold, and the actual value is illustrated by two trial crushings of 28 and 24 tons, which yielded 1 oz. and 1½ ozs. of gold respectively. The Constellation crosscut was advanced to test the reef in the belief that the absence of values in the Golden Pyke was no indication of the presence or absence of values in the Constellation.

The eastern side of the cap stone was first met in the Constellation crosscut, and showed well-laminated quartz, in which a little pyrite and blende was noticed. A little gold was seen in advancing through the reef, and the first crushing of 32 tons from the crosscut averaged 4.6 dwt. per ton. The reef proved to be 20 ft. wide, and on reaching the west wall payable gold was discovered, and the next crushing of 101 tons, including quartz from levels on the western side, averaged 9.8 dwt. per ton. Then commenced a succession of payable yields for over 2½ years.

THE FORM OF THE REEF.

The form of the reef disclosed in the main crosscut at 723 ft. was that of an irregular-shaped cap (fig. 1) with a pronounced double arch in the hanging-wall sandstone, crossed by an east-dipping fault. While there is an east-dipping break crossing the saddle, the main fault, under which the reef forms, is west-dipping, and continues down the west back as a masked fault. Extending upwards along the west-dipping fault is the neck stone, which, though variable in height, extends more than 40 ft. over the 723-ft. crosscut. For the greater length of the reef the main fault pursues a course through the west leg, dividing it into two portions. The hanging-wall portion forms the "cab stone," consisting of laminated quartz, 6 in. to 2 ft. wide, which was firmly attached to the hanging wall. No gold was ever seen in this "cab stone," and most of it has consequently been left standing. The gold was invariably associated with laminated seams in the quartz on the foot-wall side of the fault. At the 723-ft. level the saddle is well arched, but above the 622-ft. level it is sharp

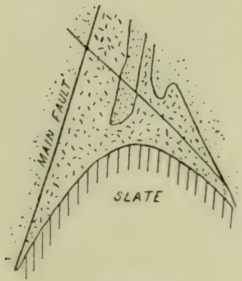


Fig. 1.
Sketch showing Constellation Reef at the 723-ft. Level.

and angular. The saddle is composed of slate, and conforms to the pitch of the strata, which is 17° south. Southwards from the 723-ft. main crosscut the pitch increases slightly, while northward, above the 622-ft. level, it flattens to 5° south.

Where intersected in the 723-ft. crosscut the reef was 20 ft. wide, including 3 ft. of sandstone, and northwards, between the 723-ft. and the 622-ft. levels, the reef maintained its size and irregular shape. A short distance below the 622-ft. level a body of spurry quartz developed on the hanging-wall side of the east leg, and the reef developed its maximum width of 32 ft. Immediately above the 622-ft. level the double-arch character disappeared, and the form changed to that of a normal saddle reef with a cap 18 ft. to 23 ft. wide and legs 6 ft. to 8 ft. thick at the turnover. The stopes along the cap were carried along for an average height of 12 ft., leaving a portion of the neck stone, which is much shorter above the 622-ft. level than above the 723-ft. level. Small variations in width, height, and shape of this saddle reef continued northwards to 675 ft. north of the shaft. Up to this point the main body of quartz had steadily risen with the pitch of the strata, but beyond it the reef commenced to dip northward, even though the strata retained their normal southerly pitch. Comparatively insignificant bodies of quartz pursued the normal track of the reef along the bedding. With this change the reef commenced to cross the bedding planes, lost its bedded form, and was transformed from a saddle reef to a large saddle-shaped spur. Within 50 ft. the change was complete, and the reef became a large spur dropping away from the west-dipping fault. The reef then rapidly decreased in size and value, dwindling eventually to a spur of small width and low values. A corresponding change occurred in the north drive on the east leg at the 622-ft. level, when the leg stone commenced to diverge from the east back. The latter continued as a well-defined track along the bedding planes, carrying a thread of quartz, while the leg stone, 15 inches wide, with colours of gold, swung in towards centre country.

Along the west leg the change brought in a large body of spurry quartz in the hanging wall, extending westwards for 20 ft., and terminating against the west leg of an overlying saddle reef. This body of quartz carried fair values for a short time.

REEF VALUES.

Table of Fortnightly Crushings.

Date.	Loads.	Oz.	Dwt.
1918.			
Sept. 24	133	56	19
Oct. 10	275	136	16
,, 22	298	167	18
Nov. 5	292	240	3
,, 19	314	254	13
Dec. 3	345	228	9
,, 17	341	356	—
Plates	—	165	14
Dec. 31	162	192	18
1919.			
Jan. 14	566	245	4
,, 28	739	290	6
Feb. 11	593	482	2
,, 25	600	586	2
Mar. 11	598	364	1
,, 24	539	160	18
Apr. 9	214	33	15
Plates	—	406	11
Apr. 23	211	113	11
May 7	301	115	14
,, 20	424	624	19
June 3	603	711	11
,, 17	610	645	6
July 1	601	450	12
,, 15	616	359	4
,, 30	534	314	8
Aug. 12	596	416	4
,, 26	660	994	—
Sept. 9	635	1326	14
,, 23	627	1616	17
Plates	—	626	—
Oct. 7	584	994	14
,, 21	498	451	15
Nov. 4	539	400	13
,, 18	631	175	2
Dec. 2	591	648	18
,, 16	519	925	13
Plates	—	253	18
Dec. 30	287	478	6
1920.			
Jan. 13	404	680	1
,, 27	723	672	1
Feb. 10	649	616	5
,, 24	697	2274	18
Mar. 9	691	2149	7
,, 23	707	2139	7
Apr. 6	502	2227	3
Plates	—	1178	16
Apr. 20	738	1818	11
May 4	637	1440	18
,, 18	704	2350	7
June 1	737	2492	1
,, 15	705	1894	19
,, 29	749	1201	13
Plates	—	1504	—
July 13	772	1385	13
,, 27	766	1857	10
Aug. 10	425	545	11
Plates	—	515	15
Aug. 24	720	1631	6
Sept. 9	748	1175	17
,, 21	765	1520	9
Plates	—	407	9
Oct. 5	692	1449	10
,, 19	707	1039	7
Plates	—	409	—
Nov. 2	650	569	18
,, 16	738	665	5
,, 30	762	254	15
Dec. 14	778	338	9
Plates	—	499	19
Dec. 30	528	503	8
1921.			
Jan. 11	471	394	4
,, 25	648	474	2
Feb. 8	685	367	14
,, 22	806	421	12
Mar. 8	734	239	14
,, 22	619	204	3
Plates	—	466	1
Apr. 5	442	167	15
,, 19	593	212	13
May 3	737	180	8
,, 17	723	168	7
,, 31	767	213	17
June 14	611	218	10
,, 28	328	170	11
July 12	258	99	13
,, 26	320	89	2
Aug. 9	324	64	—
,, 23	201	74	18
Total	43,037	60,649	0

Trial Crushings from Constellation Reef.

Date.	Portion of Reef.	Level	Position.	Yield.			Average per ton.
				Tons.	Oz.	dwt.	dwt.
1918.							
Nov. 11	Neck stone	622	At main crosscut ..	21	1	16	1.7
1919.							
May 6	Neck stone	723	20 ft. N., 30 ft. over xcut	98	4	6	.9
Nov. 11	Neck stone	723	10 ft. N., 40 ft. over xcut	113	55	18	9.9
1921.							
Feb. 14	Neck stone	723	130 ft. S., 10 ft. over level	73	1	16	.5
1919.							
Oct. 7	Neck stone	773	100 ft. S. rise to 723 ft. ..	48	2	6	1.0
Oct. 7	Cap and west leg	814	180 ft. S. rise to 60 ft. above level	75	21	13	5.7
1921.							
Feb. 14	Neck stone	814	190 ft. S., 50 ft. over level	39	4	9	2.5
May ..	East leg ..	723	100 ft. N., in level ..	66	5	13	1.7

In the past the gold production from many of Bendigo's best reefs has been regulated by a judicious mixing of high-grade and low-grade quartz, so that a steady weekly yield could be maintained. Such was not the case with the Constellation reef, and a study of the yields reflects the quality of the different portions of the reef as the workings advanced. In 1918 the high-grade yields were due to the good values in the west leg at the 723-ft. level, the east side being of no value. In the first quarter of 1919 the tonnage was increased by stoping the low-grade neck stone, with the result that the yields for that quarter show a lower average. In April 1919 the stopes on the cap became poor when the rich west leg, for the time, was out of reach, and the gold yields fell away. The stoping on the neck stone was then suspended, though resumed again at intervals. In the fortnight succeeding the suspension of the neck stopes the cap stopes improved, and the grade of ore recovered. As the 673-ft. intermediate level advanced the rich west leg was again reached, and high yields were the result in May and June. In August 1919 the north drive at the 622-ft. level reached the cap stone, which proved to be very high grade. Its richness is reflected in the excellent yields in August, September, and October, 1919. During the following period of lower yields, lasting till December 1919, the cap stone at the 622-ft. level was untouched, while the drives were extended on the east and west legs. In December 1919 the yields improved with the commencement of stopes on the west leg over the drive, while the north drive on the east leg was continued. In January 1920, these high yields were maintained by stoping the east leg over the drive while the north drive on the west leg was resumed. In February 1920, there commenced the period of phenomenal yields with the commencement of the centre stopes on the cap stone with the east and west legs above 622 ft. The maximum weekly yield was obtained on 26th May, 1920, when 1465 oz. 10 dwt. were obtained from 365 loads. The remarkable yields extended over a period of 8 months during 1920, excluding a very short period in August, when stoping was suspended during the mullocking of the stopes. In the early part of October 1920, the reef was reported to dip underfoot in the intermediate north drive on the west leg at the 600-ft. level. The change was complete by 2nd November, when the reef was reported to be dipping underfoot fast and strong 10 ft. in height and 17 ft. wide, showing a few colours of gold. A marked drop occurred in the yields with the change in prospects. Good yields, however, were obtained for some time until the exhaustion of the stopes.

The localization of values can be summarized in a general way thus :—

West Leg.—The west leg contained the highest values from the 814-ft. level up to the 622-ft. level. At the 622-ft. level it was good, and payable gold was reported in the face of the north drive at the 622-ft. level for the greater part of this length. Above the 622-ft. level it was very rich from 6 to 8 ft. below the turnover.

Cap Stone.—From 814 ft. up to the 622-ft. level the cap stone was payable. Above the 622-ft. level it formed the richest portion of the reef up to the time that the reef commenced to lose its character as a saddle reef.

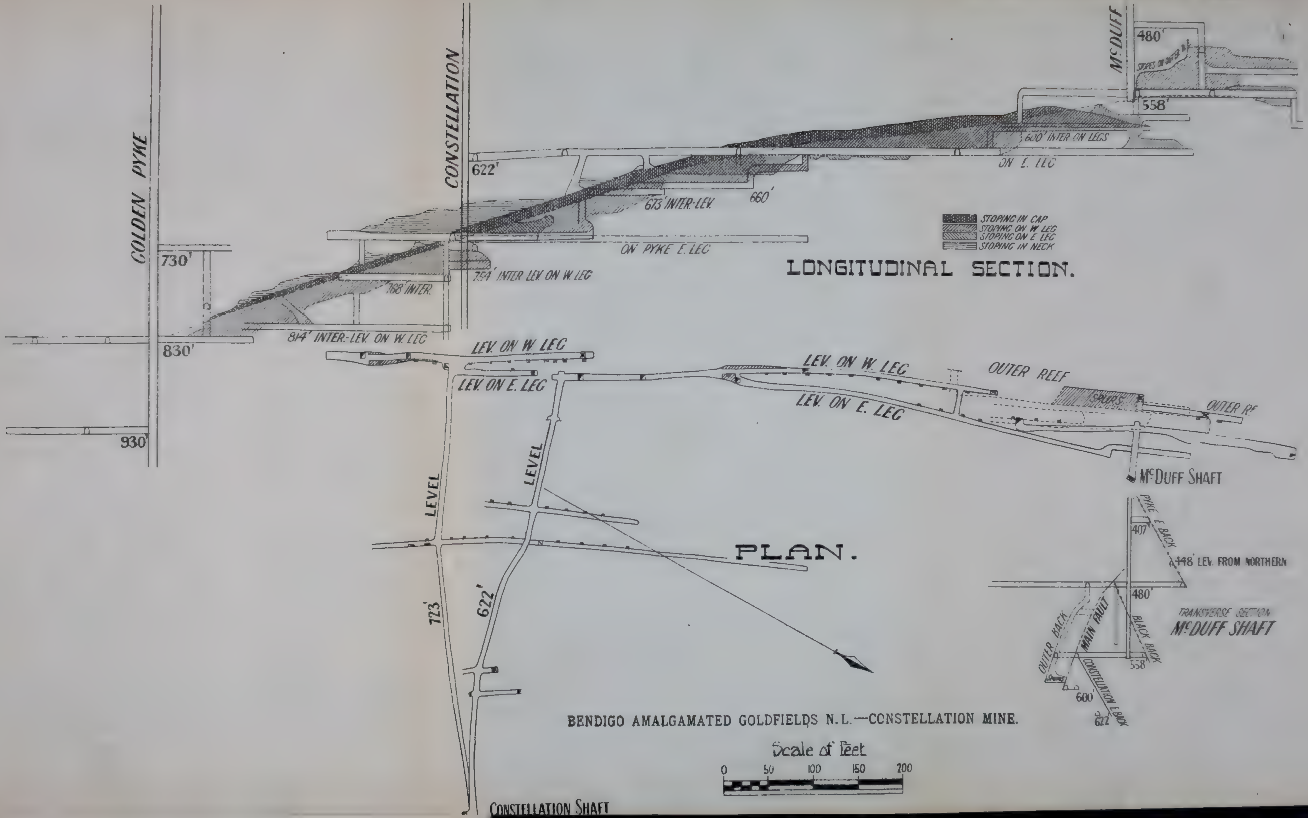
East Leg.—Below the 673-ft. level the east leg did not show any payable gold, and its value at the 723-ft. level is indicated by a trial crushing, which averaged 1.7 dwt. per ton. In the north drive at the 622-ft. level, on the east leg, payable gold was reported for the greater portion of the drive, and high values were obtained in stoping upwards. The high values were, however, not so consistent as on the west side.

Neck Stone.—The record of trial crushings from the neck stone at the 723-ft. level discloses the general low-grade character of this part of the reef. The plan shows that a fair amount of stoping was carried out on the neck stone above 723-ft. level, but the payable result of one trial crushing was not maintained. The gold content of the neck stone proved to be a few isolated specks.

THE GOLD SHOOT.

The occurrence of the gold in the reef has been similar to that described in other Bendigo reefs. In the legs and cap the gold was seen along the course of laminated seams of carbonaceous material. but in the rich portions scattered colours of gold were seen in the white quartz across the full width of the reef. In the cap the gold was mostly confined to the footwall portion of the reef; but in the richest parts the gold would extend higher above the saddle. The mineral associations with the gold were of the usual Bendigo type. Pyrite, arsenopyrite, pyrrhotite, blende and galena were common, and of these the last three were chiefly confined to the precincts of the gold. Ankerite occurred along some of the carbonaceous seams, but was at times abundant in the hanging-wall " cab stone." In the lower limits of the west leg, more or less marking the lower limits of the gold shoot below the 622-ft. level, was a highly mineralized seam containing rich gold. The chief mineral was pyrite, though the other minerals were also present. In some cases the gold surrounded the pyrite, and in other cases the pyrite and arsenopyrite crystals contained nuclei of gold.

Throughout the length of the reef cross-spurs frequently " made " from the reef into the footwall, and occasionally into the hanging-wall. In driving the 622-ft. north drive on the west leg, several of these spurs were passed. At one period special attention was directed to one spur, because it marked the point up to which gold had been seen in the leg and beyond which no gold had been seen. In addition to the disappearance of the gold, the leg stone changed its character from a nicely-laminated gold-bearing quartz to a white quartz with a streak of slate down the middle of it. Being a change from apparent wealth to apparent poverty, it caused discussion, and the 6-in. cross-spur received a measure of blame. Previous cross-spurs had not altered the character of the stone, but it would be difficult to see how such a rich gold concentration could possibly end in this manner.



BENDIGO AMALGAMATED GOLDFIELDS N.L.—CONSTELLATION MINE.

Subsequent events proved that the leg stone in another 30 ft. of driving regained its laminated and gold-bearing character, that the shoot of gold along the saddle was unaffected, and that the east leg (a few feet ahead of the cross-spur on the west leg) commenced to show high values. The drive on the east leg had passed through a period of comparatively low values, and beyond this intersection with a spur there came high values. It is possible to believe that an intersection with the cross-spurs changed the locus of gold-concentration to different parts of the reef. Further, the presence of the unreplaced slate and the absence of the distributed carbonaceous residues would account for the temporary dislocation of the gold shoot.

Considered as a whole, the view previously advanced * is maintained—namely, that the Constellation gold shoot is the continuation of the Golden Pyke gold shoot, which occurred in the east leg of the Golden Pyke saddle reef. The fault, under which the Constellation reef formed, intersects in its upward course the Golden Pyke east back and this intersection formed the main locus of enrichment in the Golden Pyke mine. In the Constellation mine the fault passes down the west leg, which, on the whole, was the richest portion of the reef. Where the Golden Pyke reef is richest, the Constellation reef is poor; where the Constellation reef is richest the Golden Pyke reef is poor. In the intervening and overlapping portion where both reefs have been worked, both have been a little more than payable. The locus of concentration of the upward circulating solutions along the west-dipping fault is in centre country in the Constellation and in the east-dipping beds in the Golden Pyke.

Northward from the Constellation, between the M'Duff and Virginia shafts, a distance of 2700 ft., the continuation of the gold shoot is unknown—it may or may not exist; but northward from the Virginia, through the Specimen Hill, New Argus, South New Moon, and New Moon leases, it has been mined with immense profits, and was known as the South New Moon reef.

The Continuity of Gold Shoots.

The continuity of gold shoots such as the Constellation and Golden Pyke, even though the stratigraphical locus is variable, is an important

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* F. L. Stillwell. "The Factors Influencing Gold Deposition in the Bendigo Goldfield." *Com. Adv. Counc. of Sci. and Ind. Bull.* No. 16, 1918, p. 23.

consideration, and can be further illustrated. Another example is afforded by the quartz deposits associated with the Victory fault, on the Garden Gully line of reef. The formation, known as the Victory spurs, occurs on the eastern limb of the anticline around the intersection of the Victory east back and the west-dipping Victory fault. The spurs have been mined from the Unity, Carlisle, Cornish, and Koch's shafts, and the distances between these shafts along the line of reef are :—

Unity to Carlisle	1170 ft.
Carlisle to Cornish	1370 ,,
Cornish to Koch's	730 ,,
Koch's to Windmill	1220 ,,

This important formation has produced gold to the extent indicated by the following approximate figures :—

Unity (up to May, 1921) ..	56,244 tons	—15,178 oz.
Carlisle (up to May, 1921) ..	185,502 ,,	—58,667 ,,
Cornish (up to May, 1921) ..	69,466 ,,	—24,908 ,,
Koch's	127,945 ,,	—38,926 ,,

The position and depth of the formation is shown on the accompanying portions of the transverse sections of these mines (figs. 2, 3, 4 and 5). In the Cornish mine the main Victory fault is accom-

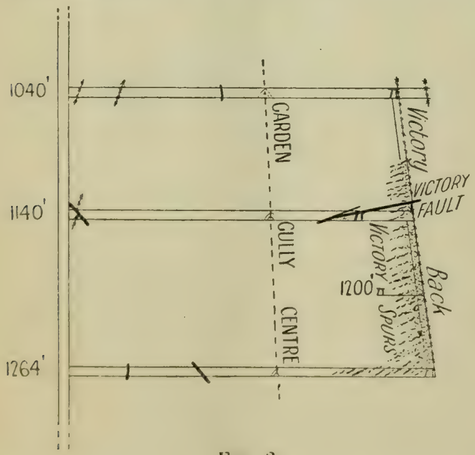

Fig. 2.

Portion of Transverse Section through the Main Shaft, Carlisle Mine.

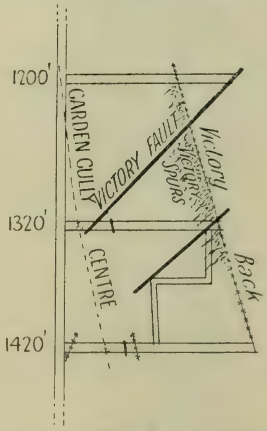

Fig. 3.

Portion of Transverse Section through the Main Shaft, Cornish Mine.

panied by a second fault, while in the Koch's mine each of these splits into two branches. Southward from the Carlisle mine there is another branch that appears in the 1264-ft. south level. The bulk of the gold from the Carlisle has come from south of the shaft from rich spurs which drop away from the Victory back. Each individual spur dips southward and persists downward, away from the main fault, to the 1255-ft. level in the Unity, where most of the gold on this run has been obtained in the Unity shaft. There remains an extensive length of this formation between the Carlisle and Cornish shafts which has yielded little gold, but which has only been prospected in the immediate vicinity of the Victory fault.

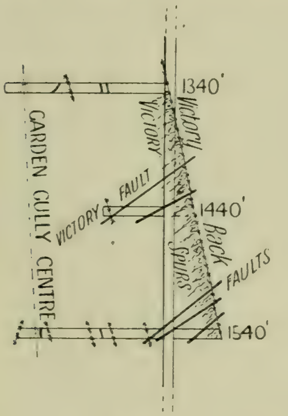

Fig. ´
Portion of Transverse Section through the Main Shaft, Koch's Mine.

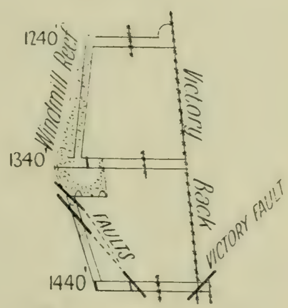

Fig. 5.
Transverse Section through Branch Crosscuts, 600-ft. North, Koch's Mine.

The probability has been previously indicated * that, southward in the Garden Gully mine, the quartz deposits, associated with the Victory fault, appear as rich saddle reefs—viz., Fiedler's reef and the rich 1447-ft. reef. Here the fault divides, and each branch forms the west back of these saddle reefs. This occurs after the fault has crossed the pitch dome of the Garden Gully anticline. These saddle reefs then persist along the southern slope of the pitch line, and have been mined by the Sea, South Garden, Ulster United, and Great Southern companies.

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* *Op. cit.*, *Bull.* No. 16, p. 23.

From the Carlisle to the Koch's the formation maintains a steady northerly pitch with the strike faults. North of the Koch's shaft the spurs, when traced along the pitch of the faults below the 1540-ft. level, eventually become both small and poor. Concurrently with the diminution of quartz in this direction the large quartz deposits of the Windmill reef developed in centre country, and have yielded an important gold shoot. From the Windmill ground 37,108 tons yielded 17,992 oz. The relation of this reef with the Victory fault is illustrated in the transverse section of the branch crosscuts, 600 ft. north of the Koch's shaft. Here appears a strong east-dipping fault, which is of no importance further south, and which intersects the Victory and associated faults and clearly carries the quartz and gold from the Victory spurs into the Windmill reef. No centre reef comparable with the Windmill reef exists opposite the Koch's shaft, and no spurry reef exists in the Windmill comparable in magnitude and importance to the Victory spurs opposite the Koch's shaft. A second formation appears in the Windmill associated with the Victory east back, but some distance above its intersection with the Victory fault. This formation also carried payable gold, but was of minor importance in comparison with the Windmill reef.

In a general way one can thus picture a more or less continuous gold shoot from the centre reefs in the Garden Gully, through the eastern spurs in the Unity, Carlisle, Cornish, and Koch's, and back again into a centre reef in the Windmill.

A similar relation appears to exist between the gold shoots in the Ironbark and Hercules mines, situated on the northern length of the Sheepshead anticline. The amount of gold obtained from these mines has been much less than the amount obtained from the Victory spurs along the Garden Gully anticline. In these mines two saddle reefs known as the Ironbark and Central Blue reefs have been prospected and mined, together with a third extensive body of quartz known as the Ironbark spurs. The relative position of these reefs is illustrated in the cross section at the Ironbark shaft (fig. 6), where the saddle reefs are 120 ft. apart. The Ironbark spurs are associated with the lower portion of the Ironbark east back around its intersection with the west-dipping Ironbark fault. These spurs extend westward across to the Central Blue east back, and in places continue eastward through the Ironbark, forming the Ironbark "outer" spurs. The Ironbark reef and the Ironbark spurs are connected with quartz along the Ironbark east back and the Ironbark spurs and the Central Blue reef are connected by the quartz along the Central Blue east back.

Lying south of the Ironbark mine is the Consols shaft, and the relative positions of these shafts and the old lease boundaries are shown in the longitudinal section (fig. 7). This section illustrates the pitch of these three reefs, and the payable portions are indicated by the heavy line.

The approximate production of these reefs is :—

	Ironbark Reef.		Cent. Blue Reef.		Ironbark Spurs.	
	tons.	oz.	tons.	oz.	tons.	oz.
Consols shaft	..	..	not payable.		not payable.	
Ironbark shaft	40,162	18,895	not payable.		16,000 (S. of shaft)	4920
Hercules shaft	26,570	8813	14,621 (to end of 840-ft. N. level)	7158	9724	2435

The Central Blue reef has been tested at several points between the Ironbark and Consols shafts, but it has not yielded anything of importance. This reef contained a famous gold shoot in Lansell's 180 mine and the Central Red, White, and Blue, commencing about 1550 ft. south of the Consols shafts. In the southern end of the 180 mine and in the Central Blue the Ironbark and Central Blue reefs merge into one huge body of quartz over 100 ft. in height. In the Consols and Ironbark it is a small reef, enlarging in the Hercules to a width of about 14 ft. across the saddle. Immediately north of the Hercules shaft it contained a short, rich shoot, to 240 ft. north of the shaft. Beyond this point it has been followed to 1200 ft. north of the shaft, and has been comparatively poor.

The Ironbark reef contained the largest shoot, which was mined from both the Consols and Ironbark shafts. A second shoot in this reef was mined south of the Hercules shaft. North of the Hercules shaft the reef has been proved to be poor for a distance of 750 ft.

South of the Ironbark shaft the Ironbark spurs form a large body of quartz which, though well prospected, has not been payable.

North of the Ironbark shaft and south from the Hercules they have been profitably mined, though the total amount of gold is not so great as in the case of the other two reefs. North of the Hercules shaft they have not as yet been thoroughly prospected.

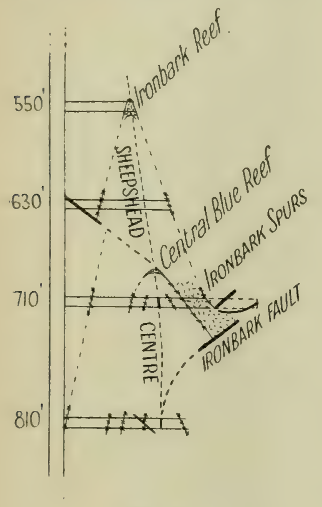

Fig. 6.
Portion of Transverse Section through the Main Shaft, Ironbark Mine.

The longitudinal section (Fig. 7) illustrates the distribution of the shoots, and indicates that the Ironbark reef is rich where the Central Blue reef is poor, and the intervening portion, where both reefs are poor; the Ironbark spurs contain their best gold shoot. These reefs broadly suggest that the gold shoot is a primary localization of a variable position, depending on the freedom and vagaries of circulation through the original network of fractures.

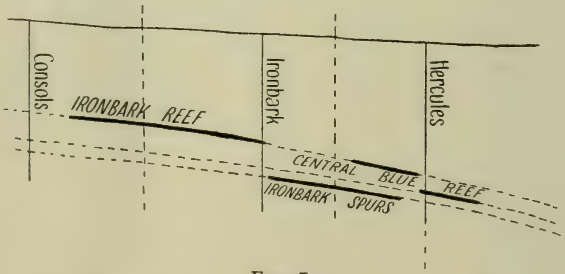

Fig. 7.
Longitudinal Section through Ironbark and Hercules Mines showing payable portions of Ironbark and Central Blue Reefs and Ironbark Spurs.

PROSPECTING.

The recognition of a variable location of the gold concentration in the Bendigo reefs has an important bearing on prospecting programmes. If a gold shoot be discovered and mined along the course of the reef, sooner or later it dies out, giving way to barren quartz.

The extension of levels through the barren quartz may chance upon the discovery of more gold, but in the majority of cases becomes dead work. The indefinite continuance of such drives, even though they may be inclined along a pitching saddle, has doubtful value, in view of the "jumps" from one reef to another reef that occur in the general continuity of a gold shoot. After the exhaustion of the Constellation gold shoot, an inclined winze, for example, from the Constellation southward along the course of the Constellation reef, would never succeed in locating the gold shoot in the Golden Pyke east back.

From time to time there has been considerable discussion on the relative prospecting values of shaft sinking and cross-cutting as opposed to the driving of levels. The disparity in cost frequently results in reliance being placed upon the driving of levels. When a general longitudinal section of the working of the Garden Gully line is studied, it is found that in many cases the shafts are well spaced, being in places 1500 ft. to 2000 ft. apart. In some cases reefs have been stoped through from shaft to shaft, but the broad fact is evident that stoping is in general confined to the precincts of each shaft, with areas of unproductive ground between shafts. It has been argued that it is reasonable to suppose that payable gold, in so many instances, is not confined to the chance spot chosen for the shaft in the early days, and that payable shoots should exist in the intervening unprospected ground between shafts. In consequence, it has been claimed that the extension of levels from existing shafts should result in discoveries. Such a policy, however, has not met with any distinct success, and the reason may lie in the variations in position of the gold shoot. The gold localization may leave the reef along which the level is driven, and then there is nothing to determine whether a policy of winzing, rising, or cross-cutting should be followed. If there should be any guide, such as a fault, the objective may be well out of reach. When such ground is penetrated by a series of crosscuts from a shaft the chances of locating gold are much greater.

In the earlier days of Bendigo's mining prosperity, when the leases were small and shaft sinking not so costly, the policy of shaft sinking and cross-cutting was dominant. In later years it was realized that, when shafts 150 ft. apart reached a depth of 1000 ft. or more and shafts 400 ft. apart reached a depth of over 3000 ft., the shaft-sinking policy had been carried to excess and beyond the limits of economical prospecting. As a result, it has been urged that, as it is possible to work and stope a reef 1000 ft. in each direction from a shaft, the expensive shafts could be most economically separated by distances of 2000 ft. This opinion has been based upon factors involved in the working of a payable reef—a problem which is comparatively simple in Bendigo, and always subordinate to the problem of finding the payable reef. The best spacing of new shafts should therefore be determined by the more important prospecting factors rather than by the mining conditions.

The main success of Bendigo mining has been derived from the occasional discovery of large rich shoots of the type of the Constellation reef, from which £338,216 worth of gold was obtained from a length of 900 ft. Other important shoots have only been 600 ft. long. Bendigo's mining problem is the location of these shoots, which can be easily missed between shafts 2000 ft. apart. New reefs in new shafts will only be tested by comparatively short levels unless colours of gold are disclosed in driving. Without this encouragement the driving of levels 200 ft. from a main crosscut would be considered a fair test of a new reef in any particular shaft. With a spacing of 2000 ft. this would mean a length of ground of 1600 ft. in which the new reef could never be prospected, though it might contain a "Constellation." The best spacing of new prospecting shafts would thus be less than 2000 ft., and should be influenced by the average length of Bendigo's best gold shoots. Economical considerations would place the shafts as far apart as possible, so that it could seem advisable to have the distances between shafts about 1000 ft. to 1200 ft.—a little more than the average length of a good shoot.

The best transverse location of a new shaft, designed to explore centre to a depth of 2000–3000 ft., was pointed out by E. J. Dunn * to be 100–150 ft. east of the anticline at surface. This determination is based on the average eastern "list" of the anticlines in depth of 10 ft. in 100 ft. The "list" of the anticlines is variable, and can only be determined after the shaft is sunk. In view of the greater quantity of quartz known to occur in the eastern limbs of the anticlines as against the western limbs,† it can be considered advisable, in fixing the position of a new shaft, to risk longer crosscuts through the upper east-dipping beds than through the lower west-dipping beds.

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* E. J. Dunn. "Reports on the Bendigo Goldfield." *Spec. Repts Dept. Mines, Vic.*, 1896, p. 18.

† F. L. Stillwell. "Factors Influencing Gold Deposition in the Bendigo Goldfield." *Com. Adv. Counc. of Sci. and Ind. Bull.* No. 8, 1917, p. 23.

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ELECTRICAL POWER APPLICATION ON THE BROKEN HILL MINES, WITH PARTICULAR REFERENCE TO REMOTE MOTOR CONTROL AND ELECTRICAL HOISTING AT THE SOUTH MINE.

BY F. J. MARS.

IN this contribution to the Institute Proceedings particulars are furnished of various special developments in electrical power practice applied to the Broken Hill mining service. These are described in a form to meet the interest of members of the Australasian Institute of Mining and Metallurgy as mining men, and will possibly be of value to mining engineers who are not experts on electrical power applications, but who, nevertheless, are interested in such developments affecting the utility and advantages of electrical power relating to Broken Hill conditions and mining fields generally.

This paper is not intended to be a comprehensive or technical thesis on electrical power generally, as applied to the Broken Hill mines, nor is it a full detail of electrical applications in the many forms and varied duties operating on the South mine in particular.

The subject would demand a much more lengthy and technical paper to deal fully with the matter, and would be more appropriate for reading before an Institute of Electrical and Mechanical Engineers.

GENERAL.

The development and application or evolution of electrical power used for general purposes on the Broken Hill mining field has had a somewhat long and varied history.

As far back as about 1890 electrical generating plants were installed in Broken Hill for lighting purposes, and one or two more or less successful attempts to use electrical energy for purely motive power purposes were carried out about 1894.

No really serious efforts to make use of electrical energy for general motive power purposes were attempted till some years later, and some time after the successful application had been fully demonstrated, both technically and commercially, elsewhere.

Of late years many Broken Hill plants are more up-to-date, and compare favourably with mining plants in any other country.

The requirements of the Broken Hill mining operations are exceptionally favourable, and the various mines, in relation to each other, are conveniently situated, geographically, for laying down a general scheme to successfully supply the necessary power for all Broken Hill operations from one central generating station, and it is unfortunate to the vested interests of Broken Hill that such a scheme was not carried out years back, before each individual mining company had launched out and installed their separate power lay-outs, with different units, many different systems, often without any regard to standardization—which now are more difficult to combine or substitute in favour of one general system.

In this respect the writer would point out that the Broken Hill mines have some twelve power stations, and, although seven of these mines now use one standard electrical supply system, there are actually four other systems for electrical power supply in service along the line of lode, which render these more or less isolated and unable to exchange power economically or conveniently, when, from every point of view, they should be interconnected for mutual advantage and assistance.

There is no apparent reason why, some time back, a central power depot, with one general congregation of steam plant, electric generators, and air compressors, should not have been installed to supply power to all these mines for all requirements, in either the form of electrical energy or compressed air. The primary saving in capital expenditure and in the after cost of generating and delivering power would have been considerable. This has been fully demonstrated in the South African Rand mines and elsewhere.

Even at this advanced period of Broken Hill's metalliferous life it would be advisable to do something in the way of centralizing or concentrating the power production for interchange and distribution in larger groups.

At least two of the large mines have recently laid down power plants of fair size, with large modern economical steam units, and even if one general power plant for the whole line of lode could now not be agreed upon, it would probably be advisable to enlarge some of the up-to-date plants and furnish power to the adjacent mines, combined into groups.

Each of these enlarged generating stations and group of connected mines could be interconnected for exchange and division of the various power loads as demanded by the individual mines or group of mines, and would go a long way towards reducing fuel and labour costs.

Considering the probable duration of the Broken Hill field, estimated from the visible bodies of ore (some 15,000,000 tons), it might be well to consider some such concentration of power plants, even with some considerable capital expenditure on a few larger modern generating units, to displace the many small uneconomical older units in general use at each mine.

With the higher cost of labour, deeper mining; probably harder and lower-grade ore to deal with, the demands for economy each year become more pressing, and power economy is now one of prime importance, accentuated by the late heavy rise in price of fuel. This, on a field so far inland, where fuel freight is so heavy, calls for every effort in the way of reducing fuel consumption. Coal costs about 65s. per ton delivered at the boilers, and water 5s. per thousand gallons.

Large capacity generating plants with large units cost less per horse-power than smaller plants to purchase and instal, are more economical in fuel consumption, attendance, and maintenance, and are technically more suitable for dealing with intermittent peaky demands for large power services for various purposes, such as winding engines, large pumps, mills, and air compressors, if operated electrically. Many of these services, if operated from a comparatively small plant—such as the majority of present Broken Hill plants—have various disadvantages in regard to regulation, and extra reserve plant held more or less idle or operating on small load, kept in readiness to deal with the maximum demand from any of these comparatively large intermittent power calls, whereas, with large plants and large units, these individual maximum demands form a much smaller proportion of the total capacity of the generating plant, and, combined with a diversity of time in their maximum call when a large number of power-consuming devices are united upon one system, there is no need to keep wasteful idle-running or reserve capacity of plant to anything like the same amount as is done with a large number of small, independent power stations.

Of recent date, as a result of a shorter working week labour-log throughout the Broken Hill line of lode, the general operations and power requirements are greatly diminished at week-ends—much more so than in past years; the power required from Saturday morning till Monday morning is reduced about 30 to 1, and, as power is furnished for each mine generally from its own individual plants operating on such a small load the cost during this period per h.p. hour supplied is very high. In many instances, if the normal running mid-weekly cost per h.p. hour is, say, 1d., this cost during week-end, would be somewhere about 10d. per h.p. hour, being the result of having at each mine boiler plants, engines, generators, sometimes air compressors, with necessary auxiliaries, all operating on very uneconomical, small outputs, with a staff of attendants at each power station, whereas one moderate or even small power plant and attendants could deal with the whole Broken Hill output during these week-ends at a cost per h.p. hour not much greater than during the normal mid-weekly load period.

Taking the nine larger mines of Broken Hill, they have installed in their combined power depots some 30,000 h.p. for electrical generating, 10,000 h.p. for air-compressing, and 10,000 h.p. for winding purposes, making a total of, roundly, 50,000 h.p. of plant for power purposes, which, of course, provides for a large proportion of reserve or spare plant held by each mine. These mine-plant capacities range from 2000 to 15,000 h.p. each, and to deal with this total output there are installed 70 steam prime movers, ranging in size from 50 h.p. to 5000 h.p. each.

If all the various loads of these mines were combined into one power station with large economical units, the whole maximum load or demand, after making provision for necessary reserve plant, could be taken care of by an installed plant capacity of about 30,000 h.p., instead of the present 50,000 h.p. of plant; and with the increased efficiency of the larger units, which could be loaded more uniformly, after making allowance for a large factor of diversity and short peak loads, the maximum demand or effective peak load would probably not exceed 25,000 h.p. for all purposes. This total h.p. could be taken care of by three or four units of, say, 10,000 h.p. each, instead of the present 30 electrical-steam units, 25 air compressors, and 15 steam winding engines.

The average fuel consumption for 25 of the 30 existing Broken Hill steam-electric generators is not less than 3½ lb. of coal per h.p. hour at the generator terminals.

The steam hoisting engines' average consumption being higher, modern turbo-generators of large capacity, operating on an average load of 75 % of full load, would give a h.p. hour at the switchboard with a consumption not greater than 1½ lb. coal.

The yearly coal consumption by the Broken Hill mines, on normally full output, is approximately 110,000 tons for power purposes.

To substitute, in a moderate-sized station, three or four large modern turbo-generators and equipment for most of the numerous small units now operating, would reduce the production of power costs some £150,000 per annum, assuming coal, water, and labour to cost within 10 % below present rates prevailing in Broken Hill.

SOME FEATURES OF ELECTRICAL APPLICATION AT THE SOUTH MINE.

The electrical generating plant equipment at the Broken Hill South mine (a description of which was published in Proceedings of the Australasian Institute of Mining and Metallurgy of September 30th, 1921) consists of—

2 3750 k.v.a. steam turbo-generators at 2400 r.p.m.
1 750 k.v.a. steam turbo-generator at 1200 r.p.m.
1 500 k.v.a. steam engine-generator at 343 r.p.m.

These generate electrical energy at 575 to 600 volts, three-phase, 40-cycle, this being the system mostly used on the Broken Hill mines, and was first adopted by one of the larger mining companies converting to 40 cycles in the year 1904 or thereabouts, and later accepted by a majority of other mines to facilitate the exchange of power and plant. This also enabled a good deal of standardization and interchangeability of equipment, such as motors, starting gear, meters, etc., which has been of great importance in the matter of spares and renewals to all the mines using similar plant and system, especially as stocks of motors, etc., could not readily be obtained in Australia as required. This general standardization in regard to sizes, types, speeds, etc., of motors and other equipment is instrumental in reducing the necessary spares, etc., carried to at least 50 % of what otherwise would be required by these mines.

A 40-cycle system (decided upon in 1904 or thereabouts) was, with due consideration, given preference as being at that time most suitable for motor power for mills and general mining service. It was also suitable for lighting ; the latter forms about 3 % of the total load.

Sixty-cycle motors have inherently a much higher speed, and naturally are not so suitable for milling and mining work, generally requiring speed reductions. Twenty-five-cycle motors, on the other hand, are inherently of lower speed ; this periodicity is not so suitable or lighting purposes, and 25-cycle generators have a maximum speed of 1500 r.p.m., which is a disadvantage for direct-connecting to moderate-size steam turbines, as both the efficiency and cost are impaired at this low speed. Furthermore, electrical equipment of 40 periodicity would give satisfactory operation on a 50-cycle system generally, and to a much greater extent than *vice versa*. This applies chiefly to motors, meters, transformers, and such appliances.

The following illustrations show some of the power-plant equipment (for further details see publication, new series, No. 43, of Australasian Institute of Mining and Metallurgy, Sept. 30, 1921—"Description of New Plant at South Mine," by W. E. Wainwright and J. C. Cunningham) :—

Plate I., Fig. 1 is a view inside the South mine power-house, showing the two 3750 k.v.a. steam turbo-generators, also the 500 k.v.a. steam reciprocating engine-generator. The latter is mostly used during light-load periods, such as week-ends and holidays.

Plate I., Fig. 2 and Plate II., Fig. 1 are views of the main power-house switchboard, showing front and back of board. These panels contain all the necessary instruments and control gear for operating the generators, feeders, and exciters.

Plate II., Fig. 2 is a view through the end of main oil-switch and bus-bar special switchroom in basement.

Plate III., Figs. 1 and 2 are views of the 5000 amp. main generator oil-switches and of the 2000 amp. main feeder oil-switches, with the heavy busbars and connections. There is not enough open space to get a better view of this switch compartment and gear. These views were taken during construction, before completion of switch gear.

Plate IV., Fig. 1 is a view taken overhead of busses and switch-gear in main switch compartment, showing incoming conductors from generators and outgoing copper feeders.

Motors.—At present the South mine has installed and connected over 150 motors, ranging from 1½ h.p. to 440 h.p. each, and aggregating some 5720 total rated h.p., although, after making allowance for under-loaded motors and motors not working simultaneously, the actual peak load demanded does not exceed 3500 h.p.

The following list gives the number and sizes of motors in general operation. These are all alternating current, 3-phase, 40-cycle, 550-volt, induction type :—

M.	H.P.		H.P.
10	1½	=	15
17	5	=	85
34	7½	=	255
42	15	=	630
6	25	=	150
13	35	=	455
5	50	=	250
14	75	=	1050
5	130	=	650
2	200	=	400
3	300	=	900
2	440	=	880
153			5720

Total h.p., 5720.

Plate IV., Fig. 2 is a view of two 440 h.p. motors, direct-coupled to and driving mine air compressors at 260 r.p.m.; capacity, 2500 cub. ft. free air per min. to 90 lb. per sq. in. each.

Plate V., Fig. 1 is a 75 h.p. motor, 480 r.p.m., belt-driving a set of grinding rolls, size 18 in. x 36 in., in crude ore mill.

Plate V., Fig. 2 shows a line of three 15 h.p., 800 r.p.m. motors, belt-driving centrifugal pumps delivering leady slimes.

Plate VI., Fig. 1 is a view of a 75 h.p. squirrel-cage motor, geared to an under-ground pump, delivering water against 1000 ft. head.

Plate VI., Fig. 2 shows the automatic float-switch starting gear for a 75 h.p. underground pump motor.

With the exception of the 440 h.p. motors and a few large motors requiring a variable speed control, such as called for on lift or hoisting duty (these having polar-wound rotors), practically all motors have short-circuited or squirrel-cage rotors, even up to 250 h.p. sizes, which were decided upon originally from various reasons, including the favourable characteristics of this type of motor.

The objections to the squirrel-cage motor (if not required for variable speed) chiefly are :—The heavy starting current required to furnish a given starting torque, being somewhat about 400 % of full load current to give full load starting effort. This heavy starting current generally has a serious adverse effect upon the voltage regulation of an electric supply system—firstly, because of the large voltage drop due to the resistance and reactance of the feeder conductors of the system which carry these heavy short-duration starting currents; secondly, because these starting currents have a much more disturbing effect upon the inherent voltage regulation of transformers and electric generators than an equal amount of current taken by a motor operating on load and speed.

These variable voltage conditions (if not excessive) are not very objectionable for purely motor-power purposes, but are disastrous to satisfactory lighting requirements, and would not be tolerated on a public supply system, which often is primarily a lighting supply utility; and, further, a public supply system usually has to transmit the electrical energy for power and lighting some considerable distance from the source of supply, which magnifies the serious result of voltage drop, due to the resistance, etc., of long transmission lines and heavy momentary motor-starting currents drawn by squirrel-cage type of motors.

These disadvantages can partly be overcome by some system of automatic voltage regulation, and with a large generating plant in operation the inherent disturbing effect on the generators of motor-starting current would not be very noticeable, but with a small plant these large squirrel-cage motors with large objectionable starting currents would not be permitted, and long public supply mains used for lighting would be seriously disturbed.

As an offset against these objections are the following:—A mine supply system of any magnitude is usually and primarily a motor power supply for their own requirements. Lighting is seldom 5 % of the total load, and moderate voltage variations are permissible. To overcome the line voltage disturbances, the writer has adopted the practice of having the whole of the lighting fed by a separate and independent set of mains from the generating busbars to the lighting transformers, thus being free from any voltage variations due to the motor feeders, starting current line drops, and such disturbances. A further advantage of having all the lighting on a separate feeder direct to the power station is, in emergency, the lighting is not cut off when the power feeders are opened from any cause, and, again, the lighting can be maintained by transference to an auxiliary or alternative source of power in case of trouble, or when desired for other reasons. The power required on a mine is usually within a much closer radius or area than with a public supply system, therefore the line drops are not nearly so great, and the South mine system of motor control, etc., as described later, practically overcomes the bad effect of such heavy line drop disturbances. The voltage disturbance caused by the motor-starting current upon the inherent regulation of the generators is taken care of by an automatic voltage regulator, such as the " Tirrill regulator," which takes care of variable loads, variable speeds, and other variable voltage disturbances, practically without any attention. So that, as far as a mine electric power system is concerned, the objectionable features of the moderate-sized squirrel-cage motor are neutralized almost completely.

Some of the advantages of the squirrel-cage motor for general mining and milling service are :—It has no slip rings, brushes, or moving contacts; experience has shown that 80 % of the minor troubles, repairs, interruption of service, etc., with motors are due to control contacts and brush-gear derangement, especially in dusty locations such as mining plants. The rotor has no windings in the ordinary sense, no insulation, no moving contacts, no high voltages—in fact, nothing to go out of order except two self-oiling bearings similar to an ordinary metal pulley, and it could almost be used as a " garden roller " without damage or affecting its function as part of the motor afterwards. The requirements for starting are simpler than with other types, and the design of starting gear available is much more rugged and suitable for everyday service under mining conditions. The squirrel-cage motor lends itself more readily for starting from a distance, and also for grouped remote motor control, as extensively used at the South mine, and described later.

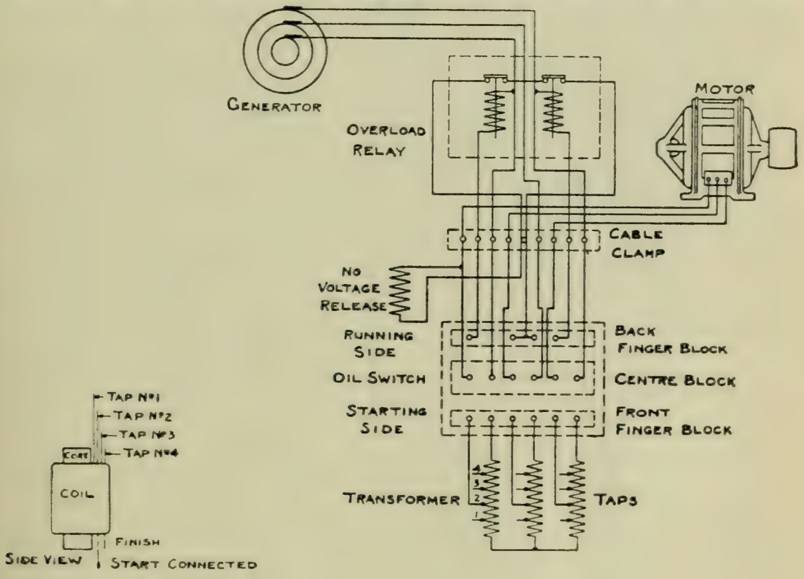
Control of 3-Phase Squirrel-cage Motors (Mining and Milling Service). —This generally consists of starting, accelerating up to speed, stopping, and in some cases reversing the direction of rotation.

As previously stated, to start a squirrel-cage induction motor on full load torque, with full voltage applied, would call for over four times full load current.

Milling and general mining motors, excepting hoisting motors and such like, are mostly daily continuously operating at full speed. They do not require speed control, and only need starting or stopping at wide intervals, especially the larger motors of the squirrel-cage type. They usually have to start up against less than full load running torque, and can exert this necessary starting effort without requiring the usually full heavy starting currents, which practically dispels any objection to squirrel-cage types for most mining services.

Small motors, up to, say, 5 or 6 h.p., may be started by simply connecting to the lines at full voltage, without objection or trouble.

For large motors the reduced starting torque and reduced starting current can be obtained by applying a lower voltage to the terminals of the motor windings; this can conveniently be accomplished as follows:—



Connections of Three-phase Auto Transformer on Starting Compensator with Low-Voltage Release and Overload Relays

Fig. 1.

(1) By impressing a reduced voltage on the motor stator windings by means of a transformer or starting-compensator; usually, the latter is designed with various taps and connections taken from its windings to give any desired percentage of full voltage to furnish the necessary maximum starting torque demanded (see Fig. 1).

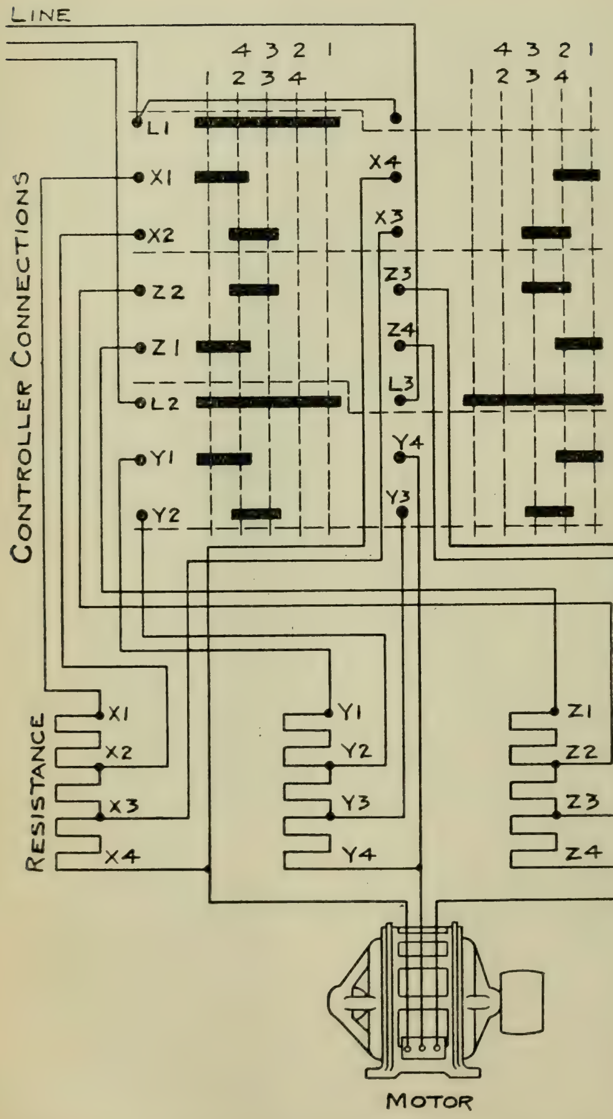


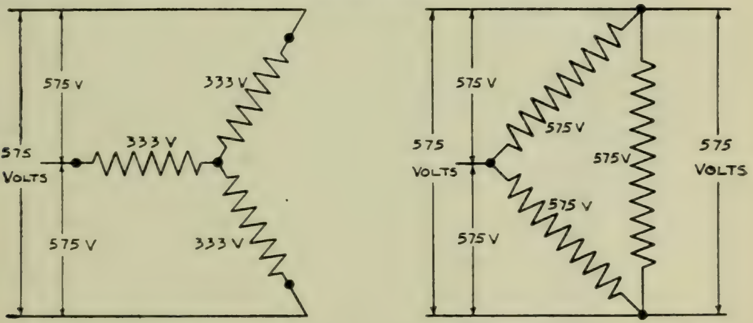
FIG. 2.

CONNECTIONS—RHEOSTATIC STARTING FOR SQUIRREL-CAGE MOTOR (Three-phase).

(2) Moderate-sized motors (5 to 20 h.p.) may be started on the rheostatic method (see Fig. 2) by simply inserting non-inductive resistance in series with the motor stator at starting to reduce the heavy rush of current, this resistance being cut out as the motor accelerates.

(3) By temporary regrouping of the stator windings at the motor by a simple change-over switching device, to so connect the windings from "delta to star" at starting (see Fig. 3), which will give an effective applied voltage approxmately equal to 60 % of full running voltage, with reduced starting current and about 40 % of full torque.

Therefore, to start squirrel-cage motors (except in small sizes) on a constant voltage system will require the necessary equipment for each motor, to start by either the transformer method (Fig. 1), rheostatic method (Fig. 2), or star–delta method (Fig. 3).



Star Connection of Motor (Starting) having 58 % of Line Voltage Impressed at Terminals of Windings.

Delta Connection of Motor (Running) having Full Line Voltage Impressed at Terminals of Windings.

Fig. 3.

Star-Delta Connections of Motors for Starting.

All these methods require the attendance of an intelligent operator, usually at or near the motor, who is generally a certificated motor man, in accordance with the mining regulations of New South Wales.

The star–delta method of starting is not generally satisfactory, because it is not flexible enough, and only permits of adjustment to one starting voltage, which gives only 40 % of full torque, and is not so suitable as the transformer method for control from a distance.

Assuming an installation (such as some of the Broken Hill mining installations) of 150 or more motors, scattered over a radius of, say, half a mile from the power plant, and all motors subject to a call to promptly start or stop at any time, necessitating the keeping of skilled attendance within reasonably close distance at all times, to comply with these regulations would involve having some 8 or 10 motor men on each of three shifts. Even with such a large staff of motor men, prompt attention would often be delayed on account of the scattered positions of the motors. Such a drawback would greatly discount a majority of the advantages of subdividing the power requirement into comparatively small units, or substituting subdivided electrical drives in mills and workshops, etc., in place of large unit steam or other engines.

In order to meet the serious objections of, first, the excessive cost of attendants, second, the danger, inconvenience, loss of time, and neglect through delay in securing an attendant when promptly required at any specific motor, the South mine, in 1906, introduced and developed a system of feeding, starting, and controlling from a central point, motors of any size and number, under the remote control of one competent motor man, who, from practically every consideration of safety, economy, efficiency, and reliability of service, etc., can attend to the control of 100 or more motors more satisfactorily than he could previously attend to a dozen similar motors.

The conditions of service for electrical power on the Broken Hill mining field, common to most mining fields, are very severe. Most of the motors and control gear, excepting the large power units, are scattered around in all sorts of out-of-the-way places, often exposed to the injurious effects of milling and mining products, such as sand, slimes, water, and other mixtures. This, combined with the high rates for labour, made it primarily necessary to have this class of equipment as simple, sturdy and fool-proof as possible, and are partly the reasons for using the squirrel-cage motor wherever possible, which also admirably lends itself to South mine arrangement of a more effective and simple group control lay-out.

This lay-out consists of having motor control or starting depots selected at centrally convenient locations adjacent to the motor-served areas, such as at a point near where the ore is delivered into a concentrating mill, or near the place where usually the main mill engine is located. Sometimes the main mine power-house serves as a very convenient motor-control station.

These control depots are fed with electrical power in bulk through 500 to 1200 k.w. capacity main feeders from the source of supply, such as the generating plant main switchboard, and protected and metered in approved manner.

The many various sizes of motors are separately fed from these control depots by connecting direct to each motor by suitably arranged multiple cables, switched and controlled from these central points.

Each motor (or sometimes two or more small interconnected motors) has its starting, indicating, and control equipment, erected and grouped in large numbers, convenient for operation by one attendant. (See Plate VII., Figs. 1 and 2, and Plate VIII., typical of various such depots in service many years in Broken Hill.)

The main features of this remote control system are as follows (see Figs. 4 and 5).

Each motor can only be started by the appointed qualified man in attendance, who is always available immediately, at the control point of all motors, when required, without the delay of seeking him about the mine or works. From this fact alone, any danger from delay is reduced to a minimum, and should strongly commend itself to the workmen, the management, and the Mines Department.

At each motor is provided a code signal-ringing key, which operates a simple drop-shutter indicator in the control depot, which in turn illuminates a lamp and motor number on the control panel of the motor calling for attention. This indicator remains visible till replaced by the attendant. The code ring is simply one ring to stop, two rings to start. In practice, even with 100 motors and grouped control, no confusion arises over the signals. Should half a dozen or more motors call for attention simultaneously, they can all be dealt with within a couple of minutes. No mistake is made by the attendant as to what is required by the individual motor signals, even if the attendant happens to be absent at the moment of call, because there are only two alternatives of operation—viz., stop or start; so, obviously, a signal from a running motor must mean stop, or for a standing motor to start. There is no delay, and it has the further advantage that the attendant can only start one motor at once, thus greatly reducing the objectionable results of having heavy starting currents from a number of motors possibly starting simultaneously, where several operators are authorized to start various small groups of motors together.

At each motor is provided a simple isolating and emergency switch, which may be operated immediately and locked open if necessary by any workman in case of need. This switch immediately cuts off power to the motor, and positively protects any workmen who may be working on this motor or on plant mechanically connected to same. This motor cannot be started till this switch is reset.

Previous control equipment for individual squirrel-cage motors consisted of a panel near each motor containing the following:—

Automatic Overload Trip Oil Switch or Circuit Breaker, with adjustable delayed-time limit trips, to cut off the motor in case of fault or persistent overload (the South mine does not use fuses, for reasons explained later).

Ammeter, to indicate the current taken by the motor at any time, to safeguard against overloading, and also for approximate power estimate purposes.

Starting Compensator, or auto-transformer, to furnish the reduced voltage for the motor during starting and acceleration before connecting the motor to full voltage.

These compensator transformers at each motor for starting cost approximately £25 to £60 each, and represented 50 % of the whole cost of the control equipment.

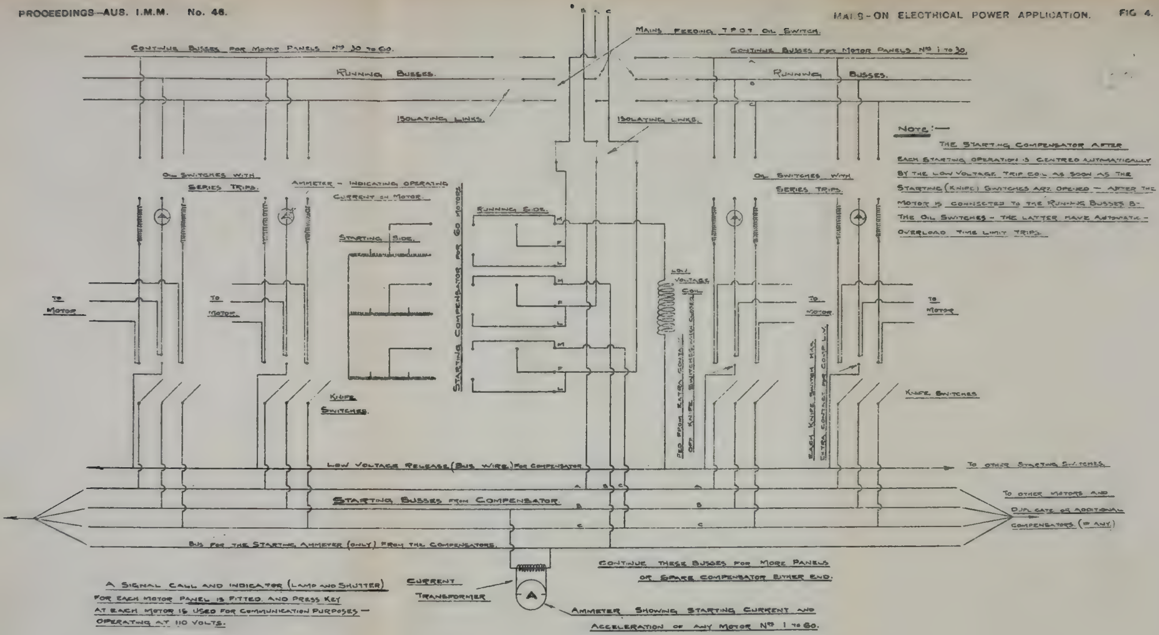
By using the central control arrangement such as hereunder described, it is found necessary to have only one starting compensator for every 25 or 30 motors, thus reducing the cost of the control gear some 40 %, as an offset against the somewhat higher cost of insulated cables feeding the motors from the central depot.

Referring to Figs. 4 and 5, also Plate VII., Figs. 1 and 2, and Plate VIII., it will be observed that each motor panel contains the following:—

An Indicating Number, illuminated from behind by an incandescent lamp, which is lighted when a call signal is given from the motor requiring attention, through the agency of a

Drop Shutter Indicator on each panel (which remains visible till reset by the attendant), operated by a signal key at the motor.

FIG 4.


A B C
Mains feeding T.P.O.T. Oil Switch.
Continue Busses for Motor Panels Nº 30 to 60.
Continue Busses for Motor Panels Nº 1 to 30.
Running Busses.
Running Busses.
A
B
C
Isolating Links.
Isolating Links.
Oil Switches with Series Trips.
Ammeter – indicating operating current on Motor.
Oil Switches with Series Trips.
Running Side.
Starting Side.
Starting Compensator for 60 Motors.
M
F
L
M
F
L
M
F
L
Low Voltage Coil
To Motor
To Motor
To Motor
To Motor
Knife Switches.
Knife Switches
Fed from extra contacts off Knife Switches when closed
Each Knife Switch has extra contact for Comp. L.V.
Low Voltage Release (Bus Wire) for Compensator.
To other Starting Switches.
Starting Busses from Compensator.
A
B
C
A
B
C
To other motors and duplicate or additional compensators. (If any.)
Bus for the Starting Ammeter (only) from the Compensators.
Current Transformer
A
Ammeter showing starting current and acceleration of any Motor Nº 1 to 60.

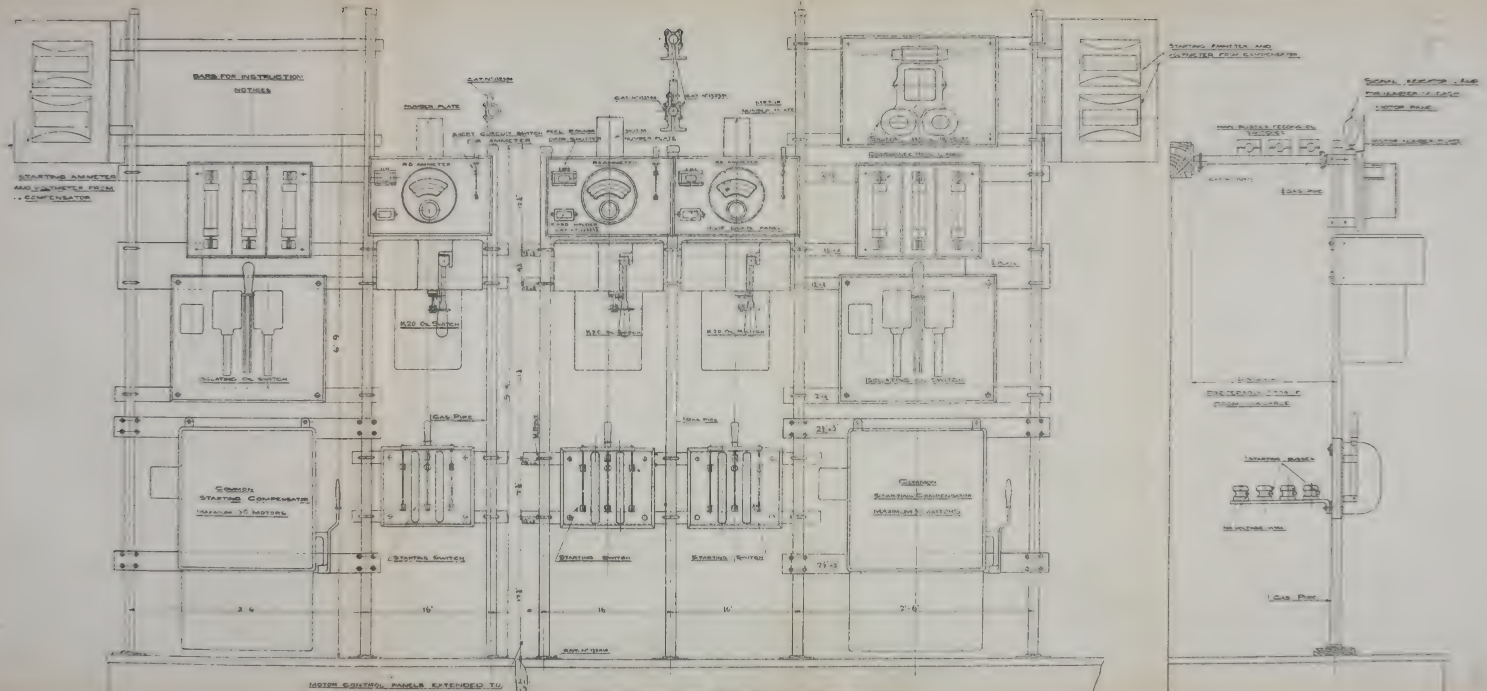
Note:—
The Starting Compensator after each starting operation is centred automatically by the low voltage trip coil as soon as the Starting (Knife) Switches are opened — after the Motor is connected to the Running Busses by the Oil Switches — the latter have Automatic-Overload Time Limit Trips.

Continue these Busses for more Panels or Spare Compensator either end.

A Signal Call and Indicator (Lamp and Shutter) for each motor panel is fitted and Press Key at each motor is used for communication purposes — operating at 110 volts.

DIAGRAM FOR SWITCHING STATION FOR REMOTE MOTOR CONTROL.

THREE PHASE ALTERNATING CURRENT SYSTEM USING SQUIRREL CAGE MOTORS, AND ONE MAIN STARTING COMPENSATOR.


BARS FOR INSTRUCTION NOTICES
STARTING AMMETER AND VOLTMETER FROM COMPENSATOR
NUMBER PLATE
CAT. No. 132394
SHORT CIRCUIT SWITCH FOR AMMETER
FEEL GONGS DROP SHUTTER
MOTOR NUMBER PLATE
CAT. No. 132396
CAT. No. 132391
NO AMMETER
AMMETER
CARD HOLDER
K20 OIL SWITCH
ISOLATING OIL SWITCH
6' 9"
5' 4"
1 GAS PIPE
COMMON STARTING COMPENSATOR MAXIMUM 30 MOTORS
STARTING SWITCH
2' 6"
16"
2' 6"
STARTING AMMETER AND VOLTMETER FROM COMPENSATOR
SIGNAL DROP AND FOR NUMBER OF CAGE MOTOR PANE
MAIN BUSBARS FEEDING OIL SWITCHES
MOTOR NUMBER PLATE
GAS PIPE
STARTING BUSES
NO VOLTAGE WIRE
1 GAS PIPE
MOTOR CONTROL PANELS EXTENDED TO ANY REQUIRED No. 16" CENTRES.

GROUP CONTROL FOR SQUIRREL CAGE MOTORS.

Ammeter, to indicate the current taken by the motor, which is entered in the log-book hourly, and, besides its utility as an indication of the motor's performance, safeguarding it against excessive or prolonged overloads, it also promptly gives evidence of any abnormal happening at the motor or the mechanical attachments connected to the motor shaft or pulley. It also serves—from the log-book time entries—to compute the power taken by the motor over a period of a day, week, month, or other period, for segregating the power costs and charges under various headings. All meters are interchangeable and identical, excepting the capacity rating.

Ammeter Switch, to short-circuit the ammeter when necessary, to relieve the moving element and jewels from excessive movement, also from abnormal current at starting, etc., thus greatly prolonging the life and accuracy of the meters.

Card Holder, containing the number, size, and location of the motor controlled from the panel.

Oil Switch or Circuit-Breaker, 3-pole, totally enclosed, of uniform type, and 200 amp. capacity. Automatic overload trip, with adjustable and delayed action to provide for short period overloads without cutting out, and low voltage trip where necessary. These breakers are connected to and take control of the motor only after starting, and when connected to full voltage. Oil crcuit-breakers are used in preference to fuses and lever switches, as being far more satisfactory, although higher in first cost. Renewals to well-designed circuit-breakers are negligible ; they can be reset instantly and often, without fear of error or danger. Fuses have to be graded to suit each size and condition of motor load. When " blown " they must be renewed by similarly-graded fuses. An excited or careless attendant may be responsible for " blowing " many fuses, with resultant cost of same, besides delay and loss from motor being stopped for much longer periods.

The South mine circuit-breakers for motors are of two standard types—one type 1 to 25 amp., the other 10 to 200 amp. Each type is identical in all features, and interchangeable, except for having different calibrated trip coils ; the latter may be changed to suit any size of motor up to 200 h.p.

Three-Pole Knife Switch, for connecting the motor (during starting and accelerating) to the auxiliary busbars, which is fed at reduced voltage from the compensator or auto-transformer when starting motors from 20 to 200 h.p. sizes. When starting the smaller motors these starting busbars can be supplied with power through suitable resistance, to reduce the heavy rush of starting current, the resistance being cut out as the motor accelerates. When the motor attains approximately full speed, full voltage is impressed upon these busbars and any attached motor—in one instance by throwing over the transformer switch to full voltage, and in the other instance (for small motors) by cutting out the inserted resistance in series with the busbars at starting. An additional small contact is arranged on each 3-pole knife switch. These contacts, when closed with the switch, feed the low voltage trip coils, and operate a holding latch on the compensators, or starting resistance switch. This provides for releasing and cutting off all starting equipment from the busbars as soon as any individual motor has been connected to the main running busbars, after closing its circuit-breaker and opening its starting switch.

The foregoing equipment is identical on each motor control-panel, and, in addition (for, say, a battery of 80 control-panels), there are three starting compensators—one for each 25 or 30 motors (one large capacity and two medium capacity) :—

One set of starting resistances and switch for small motors.

One call signal, bell, siren, or hooter, to suit location.

One voltmeter and ammeter, large capacity, showing starting current of any motor.

The back of the switchboard has two sets of busbars—one running and one for starting—which serve the whole array of control-panels. They may be sectionalized if desired, for metering or other purposes; this is usually done.

Every motor panel is connected behind, through the various previously-mentioned control apparatus, to both sets of busbars. These connections are made to serve as disconnecting links whenever necessary, by readily removing 3 bolts with a screwdriver. This isolates the panel for repairs or alterations.

As these 3 (or more) compensators provided, connect to the one common set of starting busbars, either compensator may be used for starting, and therefore act as stand-by starting equipment to all the motors; a great advantage, in case of repairs, over the individual isolated compensator method for each motor.

Operating the Motors.—When a motor in any part of the works is required to start, the workman presses on the ringing key, which is adjacent to the motor. This rings a loud bell or hooter in the control depot, releases a drop shutter on the motor control-panel required, and illuminates a lamp at the panel number. The attendant immediately responds, and

(1) Closes the starting knife switch of this panel on to the starting busbars. This also closes the low-voltage holding latches of the compensators.

(2) The operating lever on one of the compensators is thrown over to connect its reduced voltage windings to the starting busbars, thus furnishing low-voltage current which starts the motor. The starting ammeter shows the rush of starting current, and in about 10 to 20 seconds settles to normal quantity as the motor attains speed, at which point the lever is thrown sharply over, and connects full voltage to the starting busbars and motor, the latter being by now at full speed.

(3) The circuit-breaker is now closed, which connects this motor to the main running busbars, and in parallel with the starting busbars and switch, following the sequence. The starting switch is opened; this disconnects the starting busbars and resets the compensator at neutral position ready to start any other motor, and leaves the motor just started, connected to the running busbars, solely under control of its circuit-breaker. The whole operation usually takes about 25 seconds to complete, and is of the simplest routine order.

In addition to starting by the compensator reduced voltage method, small motors are connected direct on to full voltage. The rush of starting current is not objectionable, and is somewhat damped down by the steadying effect of the individual motor leads from the control depot.

For medium-sized motors a suitable set of starting resistances and controller can also be used instead of the compensators for starting motors up to 20–25 h.p., the starting busbars being fed through this resistance. As the motor accelerates, the resistance is short-circuited, and the motor transferred to its protective circuit-breaker and running busbars, similar to the other motors.

When a large number of control-panels are grouped in one room, it is preferable for convenience of the operator and time-saving to have the panels arranged around three or four sides of the room, with not more than 30 to 40 panels in one line. Every panel can thus be reached promptly without undue walking by the operator, who is usually at the log-book desk in a position near the centre of the room and panels.

Slip-Ring Motors, for starting, require the operation of inserting resistance before switching on, then withdrawing resistance from the rotor circuit as the motor reaches speed. The South mine uses this type of motor only in special cases and very heavy starting loads; then they are usually of a large size, and warrant special arrangements, which are provided, where necessary, by using electromagnetic contactor control (shown in Plate IX., Fig. 1). These are operated from the starting depot by the aid of small control or pilot wires. Fig. 7 shows a diagram of such an arrangement with connections developed for operating and controlling from the central control depot, 8 slip-ring motors, 200 h.p. each, by means of contactors and small control wires, with only one set of starting resistances. This diagram and Plate IX., Fig. 2, and Plate X., Fig. 1, make this arrangement clear; thus we are able to remote control from a central depot both slip-ring motors and squirrel-cage motors in a very simple and efficient manner.

Abuse of Instruments and Control Gear.—Where this high-class equipment is located about a mine or works in all sorts of more or less unsuitable places, often subject to inexperienced ill-use, and exposed to sand, water, slimes, acid fumes, and such, it is found that the deterioration, inaccuracy, and faults are enormously increased, which results in a greater number of stoppages, also a greater fire and human risk.

With the central control system all this sensitive equipment is installed in a suitable convenient fireproof depot, always kept in good and efficient order, and always free from unauthorized interference.

Further, the attendant is more satisfied and happy, and takes an interest in his job, even when handling 100 or more motors, where previously he was discontented and neglectful with only 10 or 12 control gears to care for when installed about the works in less favourable locations. This fact alone is of some importance with present-day labour problems.

Motor Supply Cables.—For the South mine central control, practically each motor (or small group of combined inter-connected motors) must have a set of separate suitable-sized cables feeding from its control-panel in the central control depot. This seems at first sight the chief objection and drawback, but experience has proven that this is not so. The first cost of the number of smaller cables is somewhat greater than the usual heavy common feeder, although the amount of copper involved is not greatly different. This extra cost of cables is much more than balanced in favour of the central control by the greatly reduced number and cost of the necessary starting compensators or other starting equipment.

The various sets of suitable-sized cables feeding any conveniently-situated bank or group of motors are made up into special multiple cables, often comprising 20 or more sets. Each individual cable in the multiple is of high-grade vulcanized and pure rubber insulation, usually 2500 megohms grade, or better.

These made-up large insulated multiple cables, after bunching, are thoroughly wrapped with specially prepared oiled calico and navy canvas, then varnished, which insulates and renders the whole multiple completely waterproof and protected.

The multiples are conveyed partly by steel wire suspension and partly by sheet steel or wooden ducts. They are eventually less costly and more permanent than the usual heavy (insulated) cables suspended on poles and insulators. The latter, after weathe exposure for about 18 months or 2 years, have practically lost the insulating value of the cable covering, whereas South mine experience has decidedly demonstrated that the individual cables and insulating covering inside the multiple have not deteriorated to any extent after 10 or more years' service, exposed to all weathers. This is of extreme importance in an atmosphere of sulphur or acid fumes, and the outer covering in such cases can readily be painted with a suitable medium periodically, varying from 5 to 10 years, depending upon the destructive nature of the air and fumes. Hence, at the end of, say, 8 or 10 years, the total cost of cables feeding the motors would be to the advantage of the multiple cable and central control system.

These large cable multiples are much less of an obstruction and not so unsightly as the usual system of feeders for a given large number of motors.

The individual sets of leads to one or more motors are branched out of the multiple at any desired point *en route*, and re-bunched to feed the various motors or motor where required about the works.

Reduced Fire Risk.—With a carefully laid out electrical power system the risk of fire should not be great, but, with the South mine arrangement of central control, it is reduced to a minimum. Usually, the risk of fire is with the control gear, such as switches, fuses, collecting brushes, etc., also at the various motor-feeding cables and joints, which are generally charged and under full electrical pressure at all times, close up to the motor, even when the latter is not operating. During the ordinary working hours of a mill, workshop, or other plant, any danger from fire is reduced, because attendants and workmen are then present or near by; but during non-working hours this supervision is not available, and a fire starting may go on undetected. With the South mine central control, the power to the motors, when not working, has necessarily to be always cut off at the source of the control cables, rendering an electrical fire from such cause, during idle periods, impossible, and, further, the most likely origin of fire—viz., the control gear, etc.—is all located in a suitable fireproof room, always under trained observation when alive, and not scattered about the mines or works in all sorts of places, out of constant supervision, as was, and still is, general mining practice afield.

This control, and squirrel-cage motors, would reduce electrical fires and dust explosions to a minimum in such places as flour-mills, grain storage plants, sugar and starch mills, etc.

Motor Control Signals.—Fig. 6 is a diagram of the South mine signalling system, and is practically self-explanatory. 115-volt alternating current is used. This is furnished from a small transformer, reducing current from 575 to 115 volts for signalling, also for no-voltage or press-button opening of circuit-breakers and contactors. This separate small transformer is preferable to using the 115-volt mine-lighting service, as it avoids any complications in the control circuits due to faults or grounds on the lighting network, which practically always exist on a large network to some extent.

Each control-panel is connected to the press-signal key alongside its respective motor by a small 3/22 S.W.G. wire. A common return wire of 3/20 S.W.G. joins up the whole of the returns to the control depot. This common return operates a reliable relay in the depot, which, in turn, rings a loud bell or hooter, so that a call through any signal wire rings the call bell for the attendant. Each individual signal wire is connected through a reliable drop-shutter, or relay, which closes the indicating circuit and lamp at the top of the control-panel for the motor requiring attention.

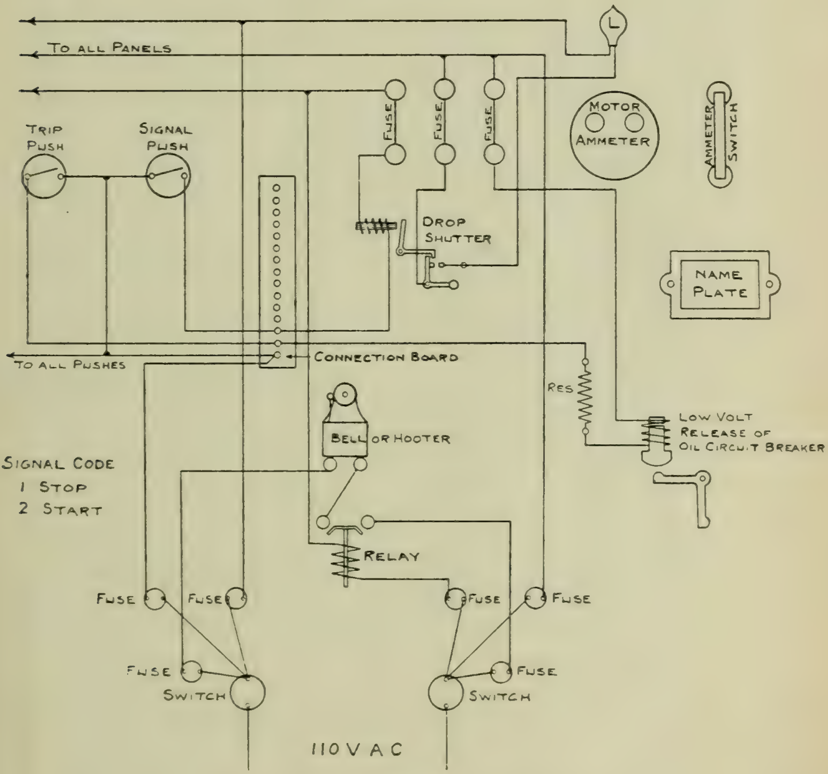


Fig. 6.

Diagram of Motor Signalling Circuit.

The whole of the necessary signal wires are made up into a separate small multiple cable and carried alongside the main multiple suspension to the motors.

The incoming signal wires to the control depot all terminate on a suitable terminal board, easy of access for locating any wire or finding a fault, or testing out.

Full Automatic Control is provided in special cases for either squirrel-cage type or slip-ring type motors. These are operated by contactors through the initial agency of variable pressure contacts for air compressors or boiler auxiliaries, and by the aid of water-float switches for pump services, etc.

Plate VI., Fig. 1 is a view of an underground quintuple plunger mining pump, geared to a 75-h.p. squirrel-cage motor, delivering water against a head of 1000 ft. A sump " water float " actuates a suitable control-switch, which, with contactors, relays, and starting compensator, automatically starts and stops the outfit at any time to comply with the water requirements of the sump underground.

Plate VI., Fig. 2 shows the actual starting lay-out for this 75-h.p. motor and pump. All the gear is located in a locked stone chamber underground, alongside the water sump.

Before starting the motor (automatically), an alarm bell is always sounded for 2 minutes to indicate that the pump is about to start.

The sump-float-pilot-switch then closes the contactors and furnishes current to the motor at reduced voltage through the starting compensator. After a definite starting time, prearranged by means of a plunger air-displacement-time-contact, the contactors then switch the full line current to the motor, which continues to operate till the sump water and float fall again to normal level.

The various pieces of control equipment can readily be identified in the photo. print (Plate VI., Fig. 2).

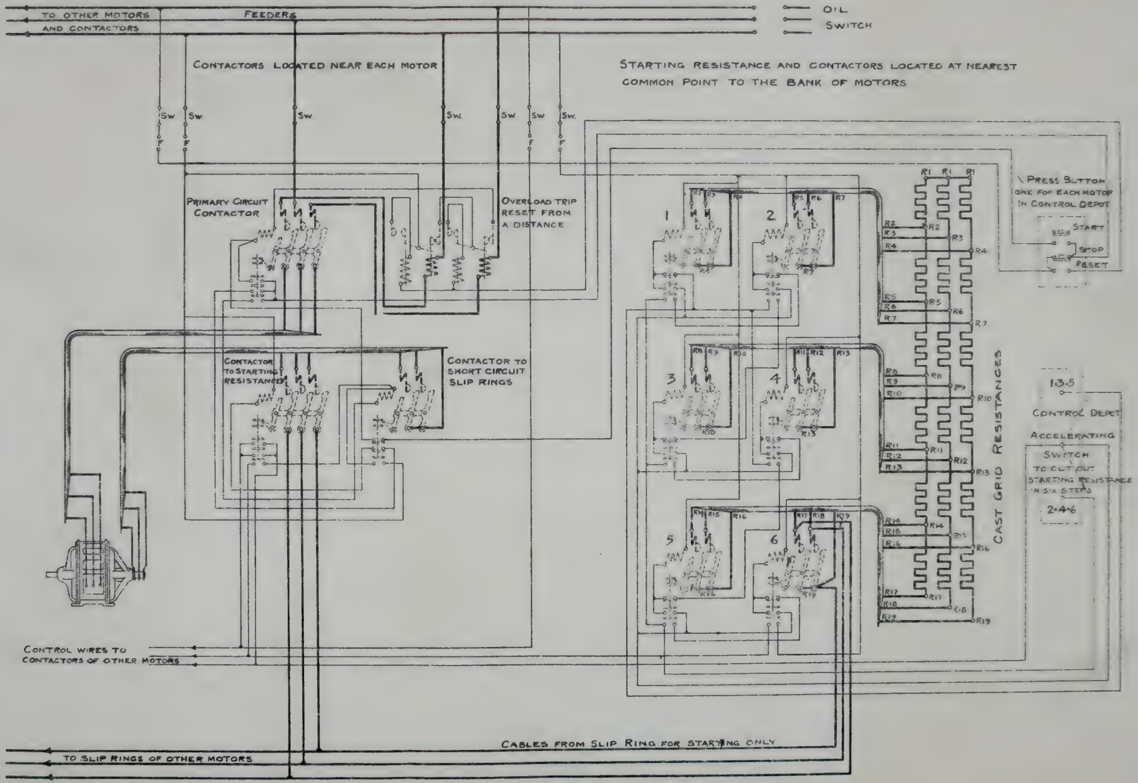
The starting compensator is of the usual hand-operated type, but the starting sequence of connections is made by the contactors automatically, although the compensator may be operated manually if required at any time.

The large air compressors (440-h.p. motors), illustrated in Plate IV., Fig. 2, which, after starting, run at full speed more or less constantly, have the air inlet controlled by an automatic throttling or unloading valve, governed by the prearranged pressure of the air system. The admission or restriction of the inlet air determines the amount of electrical power needed to operate the compressor at any time, and this is automatically taken care of by the motor without attendance other than a periodical inspection.

Remote Control of Large Motors, with wound rotors and slip rings, is sometimes required for starting against very heavy torque, similar to above 440-h.p. motors and air compressors. This is provided by

WIRING DIAGRAM FOR 200 H.P. MOTORS.

ARRANGED FOR ONE SET OF STARTING RESISTANCES WITH CONTACTORS
FOR STARTING A BANK OF EIGHT SLIP RING MOTORS


TO OTHER MOTORS AND CONTACTORS
FEEDERS
OIL SWITCH
CONTACTORS LOCATED NEAR EACH MOTOR
STARTING RESISTANCE AND CONTACTORS LOCATED AT NEAREST COMMON POINT TO THE BANK OF MOTORS
Sw
F
PRIMARY CIRCUIT CONTACTOR
OVERLOAD TRIP RESET FROM A DISTANCE
PRESS BUTTON ONE FOR EACH MOTOR IN CONTROL DEPOT
START
STOP
RESET
1
2
3
4
5
6
R1
R2
R3
R4
R5
R6
R7
R8
R9
R10
R11
R12
R13
R14
R15
R16
R17
R18
R19
CONTACTOR TO STARTING RESISTANCE
CONTACTOR TO SHORT CIRCUIT SLIP RINGS
CAST GRID RESISTANCES
1·3·5
CONTROL DEPOT
ACCELERATING SWITCH TO CUT OUT STARTING RESISTANCE IN SIX STEPS
2·4·6
CONTROL WIRES TO CONTACTORS OF OTHER MOTORS
CABLES FROM SLIP RING FOR STARTING ONLY
TO SLIP RINGS OF OTHER MOTORS

resistance in the rotor circuit of the motor, which is cut out in 6 or 7 steps by contactors, relays, etc., similar to that shown in Fig. 8, which illustrates this arrangement. The starting is controlled from the central control depot, similar to other motors. The first two resistance

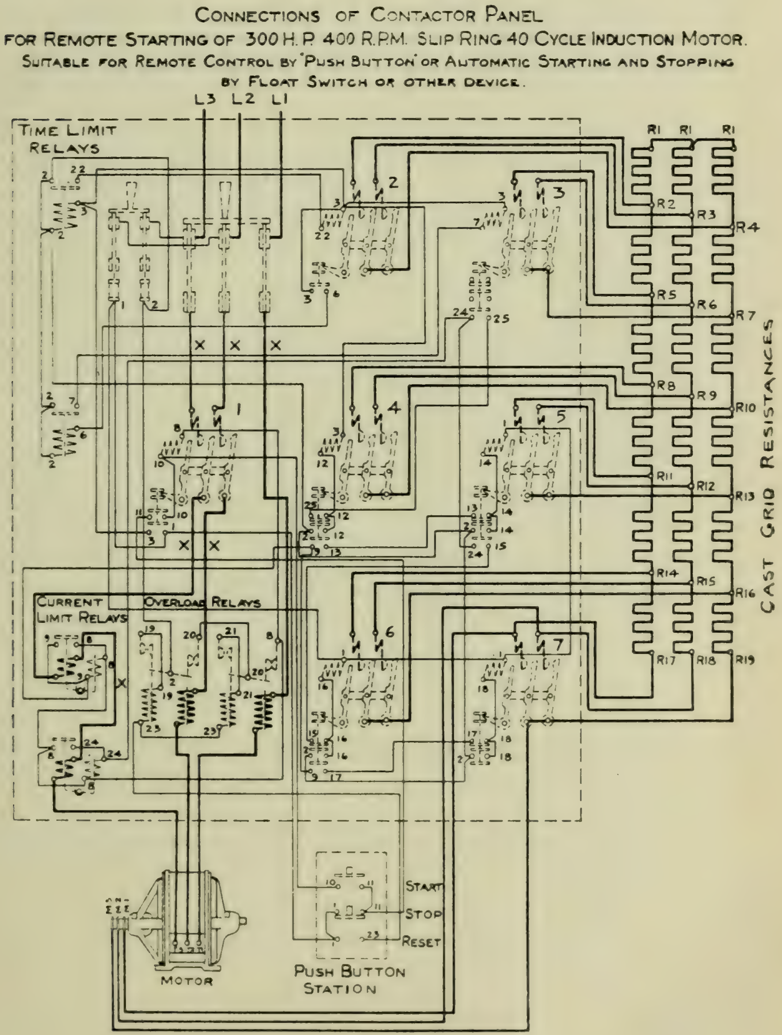

Connections of Contactor Panel
for Remote Starting of 300 H.P. 400 R.P.M. Slip Ring 40 Cycle Induction Motor.
Suitable for Remote Control by "Push Button" or Automatic Starting and Stopping by Float Switch or other Device.
L3 L2 L1
Time Limit Relays
R1 R1 R1
R2 R3 R4
R5 R6 R7
R8 R9 R10
R11 R12 R13
R14 R15 R16
R17 R18 R19
Cast Grid Resistances
Current Limit Relays
Overload Relays
1
2
3
4
5
6
7
Motor
Start
Stop
Reset
Push Button Station

Fig. 8.

steps are delayed cutting out a prearranged time after switching on by means of an air or oil plunger, relays and contacts delaying the closing of the first two resistance contactors. The remaining steps are subject to current limit relays' control. This arrangement gives almost any desired combination of time and current adjustment to cope with severe starting loads.

The motors may be started from the central motor control depot, or started automatically by a water-float, pressure-contact, or other suitable device, without an attendant.

Power Consumption of Motors.—It is usually desirable to segregate approximately, or apportion within 1 or 2 %, for mine-costing purposes, the power (in kilowatt hours or kelvins) used by the various motors fed from each control station. To instal a watt-hour meter for each motor is not advisable, and would be too costly.

The South mine determines these power proportions quite simply from its control-station log-book records, adjusted with one total-power-supply-meter for each central control station.

Fig. 9 is a copy of a log sheet covering 25 motors. Other continuation sheets provide for the remaining motors. It will be noted that several motors are not operating, due to the restricted operations on the Broken Hill mines.

Ampere readings are taken for each motor every hour, and the starting and the stopping times are "ticked on or off" at the nearest quarter-hour column ruled in the log-book.

Every 24 hours the average ampere reading and the total hours run of each motor are recorded in the log.

The total power supplied to each control station is metered at the power-house switchboard—including all losses—and logged each 24 hours.

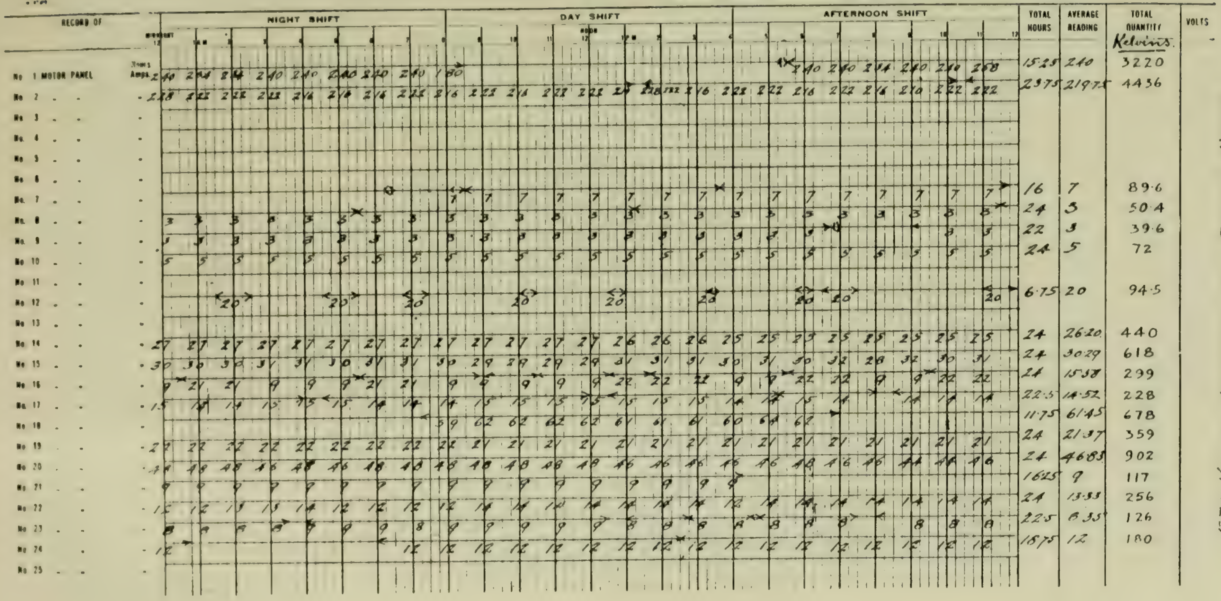
The power to each motor is estimated, from the control-station log-book, by the simple formula—

$$\frac{\text{Amperes} \times \text{volts} \times \sqrt{3} \times \text{power factor} \times \text{hours run}}{1000} = \text{kelvins.}$$

The sum of these motor records is balanced against the total power metered at the power-house switchboard, and closely agrees. Extreme accuracy is not essential. This method is found as reliable as, and better for this duty than, a number of commercial watt-hour meters.

RECORD DURING RUNNING HOURS MILL MOTOR HOUSE

FRIDAY DAY 14TH JANUARY 1921


NIGHT SHIFT
DAY SHIFT
AFTERNOON SHIFT
Volts
Amps.

RECORD OF	TOTAL HOURS	AVERAGE READING	TOTAL QUANTITY Kelvins.	VOLTS
No. 1 MOTOR PANEL	15.25	240	3220	
No. 2	23.75	219.75	4436	
No. 3				
No. 4				
No. 5				
No. 6				
No. 7	16	7	89·6	
No. 8	24	3	50·4	
No. 9	22	3	39·6	
No. 10	24	5	72	
No. 11				
No. 12	6.75	20	94·5	
No. 13				
No. 14	24	26.20	440	
No. 15	24	30.29	618	
No. 16	24	15.38	299	
No. 17	22.5	14.52	228	
No. 18	11.75	61.45	678	
No. 19	24	21.37	359	
No. 20	24	46.85	902	
No. 21	16.25	9	117	
No. 22	24	13.33	256	
No. 23	22.5	8.35	126	
No. 24	18.75	12	180	
No. 25				

575 VOLTS CONSTANT AT POWER HOUSE.

FIG. 9.

The power factor figure in the formula is obtained from a tabulated list (or curve) for the various sizes of motors at fractional loads.

The volts are taken at the power-house switchboard to include all losses.

Electric Mine Hoists (generally termed "winders" in Australian mining fields) have not been used to a great extent on the Broken Hill mines, although they have of recent years made great headway outside of Australia, and they are generally displacing large and small steam-driven mine-hoisting outfits on most of the large and important mines abroad using modern equipment.

The Broken Hill mining field as a whole offers very favourable conditions for using electric mine hoists; but electric drives have, with two or three exceptions, had little application to mine-hoisting service in these mines.

Various local factors are responsible for the backward position, in this matter, of such an important metalliferous mining field, the main retarding factor being the absence of a large general central power-station with plant to supply practically all the power, including hoisting, required for the Broken Hill mines. Such a power station could technically and economically deal with the short, heavy peaks and very intermittent load demands of large individual main-shaft hoists so difficult to cope with on small capacity stations.

The intermittent peak load requirements of the main shaft hoist would usually form such a large proportion of the total electrical load output of most of the existing individual mine electric power plants as to practically preclude electric hoisting for main shafts. Such a comparatively small generating capacity, with these load peaks, would suffer in the matter of efficiency, reliability, and operating difficulties, whereas, if the individual power plants installed on each mine had capacity capable of dealing electrically with all the required power on the mine for its mills, workshops, pumping, ventilating, air-compressing, hoisting, etc., instead of dealing with only portion of the power used, it would be feasible for each mine to hoist electrically to advantage, without difficulties arising from the intermittent nature of the power drawn by the "winders."

With one large central power-supply station, the various hoisting loads from some 15 to 20 shafts would somewhat overlap, still giving a fair diversity factor to the combined load, and the effective peak, allowing for this diversity, combined with the power used for other mining purposes, would amount to not more than 15 % of the total generating capacity operating in such a central station under suitable provisions.

This hoisting load would not in any way be difficult to technically and economically handle with the combined power loads, especially with large output modern high-speed steam turbo-generators, which have better governing qualities, larger overload capacity, and other characteristics for more efficiently dealing with intermittent peak loads than previously possessed by the retiring reciprocating type steam prime-movers of earlier practice.

Direct Steam-Engine Hoisting Plants are very inefficient in fuel consumption, and are mostly non-condensing. The actual consumption figures per shaft h.p. are difficult to obtain. Some actual test steam-consumptions given of large high-class twin hoisting engines average about 50 lb. of water per shaft h.p., compared to 24 lb. per shaft h.p. for electric hoists when drawing power from turbo-generators of moderate capacity. From which it is obvious that where power is generated and available electrically in such bulk as not to be affected adversely by the intermittent nature of the hoisting power cycle, the economy is greatly in favour of electric hoisting.

A steam engine's hoisting capacity is designed to cope with the maximum conditions at starting, and has a surplus of capacity most of the time, which leads to uneconomical fuel consumption.

The constant cooling of steam cylinders, valves, and other engine parts is responsible for heavy steam condensation, and is a serious loss of energy, especially as it is usual practice, during the idle portions of the hoisting cycle, to keep all these parts hot by means of live steam, even if the engine has to stand idle for some time between each hoist.

Many mine shafts are a considerable distance from the boilers, resulting in considerable loss in pressure and condensation of transmitted steam. Power transmitted electrically is decidedly more efficient. For shaft hoists located underground, the electric drive has practically everything in its favour.

Another factor preventing the use of electric hoisting generally—for moderate powers at least—is the capital expenditure for conversion from steam where the latter is already installed, and, although the operating costs will greatly favour the electric drive, the life of the shaft, tonnage raised, and other considerations may not warrant the cost of conversion.

For new shafts, or new hoisting plants, especially if power has to be transmitted, the many advantages of the electric hoist should be given serious consideration, and will mostly be found cheaper in first cost, operation, and maintenance.

Choice of type of motor for a mine hoist in any particular proposition must be carefully considered in conjunction with the nature and characteristics of the duty in each case. Electric supply systems for large mines are almost universally alternating current, and there is no doubt of its superiority generally for mining and milling purposes.

Electric mine hoists are practically driven by either polyphase induction motors or shunt-wound direct-current motors, each having its special qualities and field of operations. As a general rule, shunt-wound motors are only used for the largest hoisting plants, and induction motors for a lower range of powers, although their respective fields considerably overlap.

Induction Motors usually draw their power direct from the A.C. supply system, and absorb power when in actual operation *only*. Load-balancing of peaks is not so generally sought after because of the usually smaller loads dealt with compared with D.C. hoists.

D.C. Shunt Motors for large hoists naturally cannot draw power direct from the A.C. supply system, but get power through a motor converter (or some substitute) always kept running. Therefore, power, to some extent, is being absorbed while the hoisting motor is stationary. The motor-generator is also necessary with this system, for control purposes and load-balancing to smooth out the heavy load peaks, otherwise reflected back upon the main generators, engines, and boilers.

For very large powers and peak loads, the D.C. shunt hoisting motor has many advantages. With the Leonard system of control and the Ilgner flywheel, used to smooth out and compensate the power peaks and valleys of the hoisting cycle, it makes an effective device, especially for constant hoisting service duty dealing with large tonnages.

If the hoisting service demand is not regular, the constant operation of the motor-generator-flywheel set which is entailed, would use such a large proportion of the power in idle running that, on the question of efficiency, it would be advisable to consider the induction motor in one or other of its well-known reliable forms of application with or without arrangements for power balancing.

The Shunt Motor's Speed from start to full speed for a definite load may be varied in two ways as required—

(1) By varying the voltage of the D.C. current supply to the motor.

(2) By introducing variable resistance in series with the motor armature.

The latter (rheostatic control) is not so efficient a starting method as regards power economy, and is seldom used with large D.C. motors for hoisting, which often operate for appreciable time at reduced speeds, resulting in a deal of waste power in the regulating resistance. Usually, the Leonard control, which gives a variable voltage from the generator, is most suitable. This necessitates running a D.C. generator for each hoist. These generators may be driven by an A.C. induction motor (preferably) with or without the inclusion of a power-balancing flywheel, to smooth out the load peaks upon the generators, etc.

The D.C. Generator may be driven by other reliable constant source of power, such as a water-wheel or steam engine. It may sometimes be quite a favourable proposition to direct drive this special generator with one of the main generating plant prime movers, such as the steam turbines used for generating the A.C. power. This could readily be arranged by suitable shaft extension for the D.C. generator. This would reduce the size of the A.C. generator capacity otherwise necessary, and, as modern steam turbines have large overload capacity, especially for intermittent duty, as well as having considerable flywheel effort to help supply the starting energy and smooth out the peaks of power called for by the hoist motor, practically the intermittent hoisting load would be taken care of by the steam boilers, where it would be least troublesome to provide for, and would not disturb the voltage regulation of the main alternating electric system used for the other mine services.

The speed and torque of the hoist motor under this D.C. system is controlled by altering the voltage of its individual *generator*, by using resistance in the (small power) "shunt" exciting windings of the *generator*; this gives an efficient and perfectly smooth control at the motor and hoist. A separate exciter forms part of the outfit to supply the shunt-exciting-current to the hoist motor and its allied generator.

The A.C. Induction Motor Speed Control, with one or two special exceptions seldom used for hoisting, is obtained by varying the amount of resistance in series with the rotor circuit (rheostatic control), this having charcteristics similar to the rheostatic-controlled shunt motor, and similarly wasteful of power when continuing to operate at reduced speeds. Therefore, the efficiency of the induction motor during the reduced speed or control period is not so good as the voltage-controlled shunt motor; but, against the latter, has to be debited the usual constant running losses of the motor-generator set, which reduces the over-all efficiencies, especially on intermittent duty.

At full speed and load the efficiencies of the motors alone are about equal, but the first costs of each complete equipment, covering equal duty, are greatly in favour of the induction motor.

For moderate hoisting speeds of, say, from 800 ft. to 1500 ft. per minute, and depths of 800 to 1500 ft., the induction motor, if suitably proportioned to the duty, should acclerate to full speed in 4 or 5 seconds, and this speed is not too great for all usual hoisting, hence the time the motor is running at the inefficient reduced speeds is a very short portion of the complete hoisting cycle, so does not to a great extent adversely affect the overall efficiency of the hoist. Furthermore, mostly all mine hoists actually run but a small percentage of the 24 hours; even with a so-called busy mine shaft the motor is idle a greater part of the period. This reacts greatly in favour of the over-all efficiency of the induction motor hoist, especially for moderate speeds, where acceleration to full speed is quickly attained, as compared with the over-all efficiency of the D.C. shunt motor, combined with the necessary constant running motor-generator-flywheel set, whose field of operation is primarily for large high-speed heavy-duty constantly-operating hoists.

A complete general review of the various characteristics of the different motors and control for mine hoisting would involve a lengthy article to do justice to the subject; the writer therefore confines his description generally to the induction motor shaft-hoisting equipment for moderate powers, as installed some time ago in Broken Hill by the South mine. These are the only electric mine hoists so far operating on the Broken Hill field, if the writer may except a small equipment of about 90 h.p. installed for the Junction North Company at the Pinnacles mine, and using a Deisel oil engine prime-mover for the generating of electric power for the mine hoist and other requirements.

The Broken Hill South Mine uses an expansive type steam hoisting engine for its main shaft, which is situated near the main boiler plant. Conversion to electric drive for this shaft hoist has been discussed, but is not contemplated at present, due to the heavy cost of conversion with prevailing high prices of electrical and other plant, also the uncertain prospects as regards the future labour and metal position, and need of a suitable site to accommodate the electrical equipment lay-out.

Electric Hoists are used on the three other shafts on this mine. As these are practically identical with each other, a description of one will apply to the three hoisting sets.

These are illustrated by Figs. 10, 11, 12 and 13, and Plate XI.

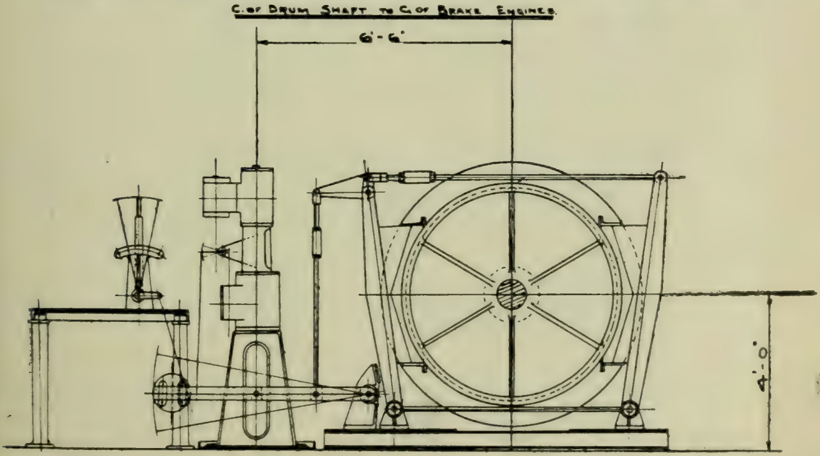


FIG. 10.
GEARED ELECTRIC MINE HOIST.

The hoist gear is the design of Mr. J. C. Cunningham, chief engineer of the Broken Hill South Ltd., and has the following features :—

Drums.—One fixed and one loose drum is provided on the shaft, the latter being clutched by a dog clutch to the shaft as required, the shaft being square where the clutch slides thereon.

The drums are plain cylindrical, without wooden lagging, the cheeks being of cast iron and the drums of steel plate.

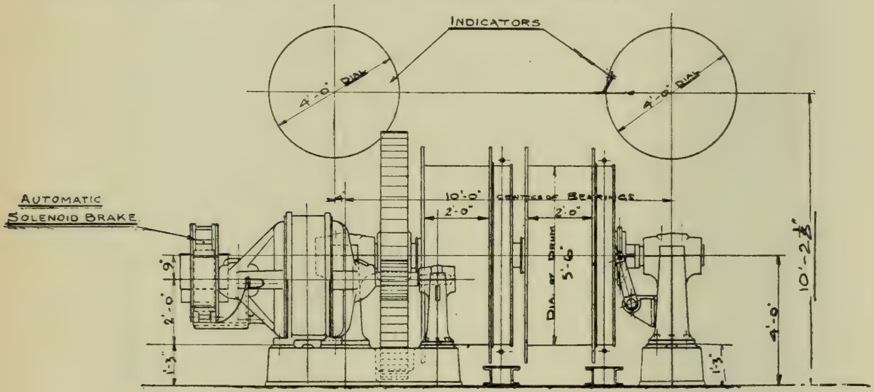


FIG. 11.

GEARED ELECTRIC MINE HOIST.

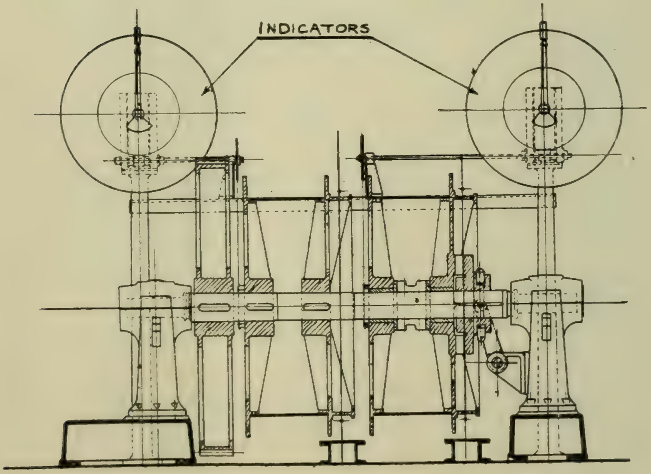


FIG. 12.

GEARED ELECTRIC MINE HOIST.

Brakes.—The brake races are cast integral with one cheek of each drum. The brakes are of the post type, formed from steel webs having cast-iron distance-pieces, wooden packing pieces, and Ferodo lining strips.

These are operated by engines using compressed air and having oil-control cylinders.

The air is used both to apply and release the brakes, and in the event of the air failing, heavy weights on the brake engine levers apply the brakes.

The brakes are operated from the control platform through jockey levers, which have the effect of causing the brake engine to move proportionally with any movement of the control handle, thus permitting gradual application of the brake.

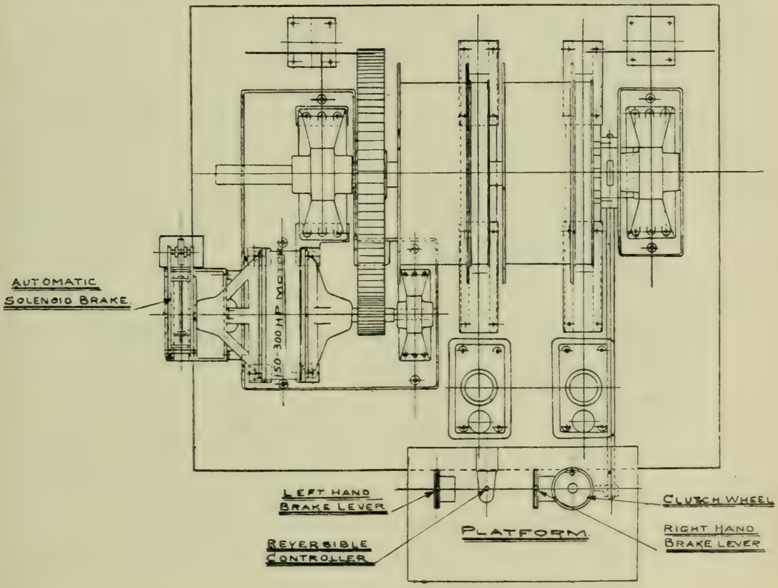


FIG. 13.
GEARED ELECTRIC MINE HOIST.

A subsidiary solenoid brake is also provided on a drum on the motor shaft, and which is instantaneously released or applied electrically.

Gearing.—The gearing used is of the plain spur, involute type, and comprises a cast-iron wheel of 112 teeth, 10-in. face, on an 89.6-in. pitch circle, which gears with a mild steel 17-tooth pinion on the motor shaft.

Speed of Hoisting.—The speed of rotation of the drum shaft is 60 r.p.m., which gives a maximum hauling speed of 1000 ft. per minute. The synchronous speed of the motor is 400 r.p.m.

With a view to raising the speed of hoisting, the drum shaft has been provided with a projecting end, so that a direct-coupled motor can be coupled to same in place of the gear drive. By this means hauling speeds, possibly as high as 3000 ft. per min., can be arranged when required.

Bearings.—All bearings are of the ring-oil, self-aligning type.

Cage Indicators.—The indicators are of the dial type, with adjustable level indicators on the edge, each of which also carries the shaft chair indicating lamp for its particular level and compartment.

The pointers are operated by worm gear and chain drive off the drums.

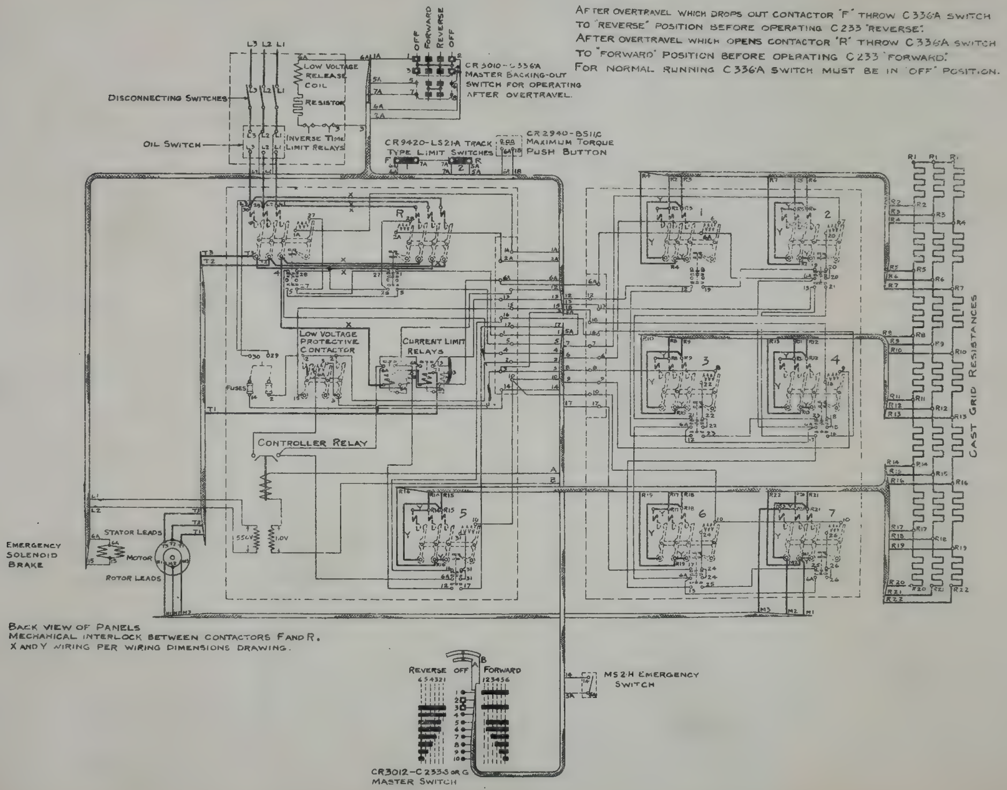
Operating Platform.—On the operating platform a seat is provided for the driver, in front of whom the following control handles are arranged from left to right :—

Left-hand drum brake.
Reversible master-controller, with "dead-man stop" contrivance, as further described ; see Fig. 14, also Plate XI.
Right-hand drum brake.
Clutch handle for operating the right-hand drum clutch.

Duty.—The duty of the hoist, as at present arranged, is to raise a 2500-lb. cage containing six men, plus 1500 ft. of rope weighing 2100 lb., in single or unbalanced gear. The total load in this extreme case would be 5500 lb., but normally the cages are always in balance, excepting when changing with an empty cage.

Hoist Motor is of the induction type, variable speed, with polar-wound rotor, collector rings, and carbon brushes for connecting to the outside control resistances. It has two main bearings with self-oiling rings, extended shaft to drive gear pinion, and outboard bearing to support the pinion drive on one end, and short shaft at the opposite end to attach brake drum for electric "solenoid brake." It is alternating current, 3-phase, 40 cycles, 550 volts, 400 r.p.m. maximum speed, 300 h.p. on intermittent rating, 40° C. temperature rise, of General Electric (Schenectady) standard motor design, and the three hoist motors are identical and interchangeable in whole or in part, with spares.

DIAGRAM OF CONNECTIONS AND APPLIANCES FOR MINE HOISTING INDUCTION MOTOR 300 H.P. ELECTRO MAGNETIC (CONTACTOR) CONTROL


L3 L2 L1
OFF FORWARD REVERSE OFF
Low Voltage Release Coil
Resistor
Disconnecting Switches
Oil Switch
Inverse Time Limit Relays
CR 3010-C 336A Master Backing-out Switch for operating after overtravel.
CR 9420-LS21A Track Type Limit Switches
F R
CR 2940-B511C Maximum Torque Push Button
R
Low Voltage Protective Contactor
Current Limit Relays
Fuses
Controller Relay
55CV
110V
Stator Leads
Emergency Solenoid Brake
Motor
Rotor Leads
1 2 3 4 5 6 7
Cast Grid Resistances
R1 R2 R3 R4 R5 R6 R7 R8 R9 R10 R11 R12 R13 R14 R15 R16 R17 R18 R19 R20 R21 R22
M1 M2 M3
T1 T2 T3
Reverse Off Forward
654321 123456
A B
MS 2-H Emergency Switch
CR3012-C 2333 or G Master Switch

After overtravel which drops out contactor "F" throw C336A switch to "reverse" position before operating C233 "reverse".
After overtravel which opens contactor "R" throw C336A switch to "forward" position before operating C233 "forward".
For normal running C336A switch must be in "off" position.

Back view of panels.
Mechanical interlock between contactors F and R.
X and Y wiring per wiring dimensions drawing.

Control Equipment.—The direction of rotation is controlled by triple-pole air-break contactors, which operate the primary (stator) circuit. There are two contactors for this purpose, electrically and mechanically interlocked to prevent both contactors being closed at the same time. One of these contactors, when closed, produces a clockwise rotation of the motor, and the other a counter-clockwise rotation (see Plate XII. and Fig. 14).

The speed of the hoist motor is controlled by varying a resistance in the secondary or rotor circuit. This resistance is of the cast grid type, and is arranged to be " cut in or out " in balanced steps on the 3-phase windings of the rotor, by means of seven contactors suitably interlocked to operate in proper sequence. All contactors are primarily controlled by a small master-controller (Plate X., fig. 2) located on the operator's platform near the hoist.

There are eight control steps in either direction. Six of these steps are capable of hand control to a limited degree.

All the steps are automatic—that is, if the controller be instantly thrown into full running position, the contactors will close in eight successive steps, but are delayed or deferred depending upon other factors when necessary.

The closing of the successive secondary (rotor) contactor steps is governed by current limiting relays which are set at a predetermined value of current, so that the steps of resistance can be cut out only after the current taken by the motor has fallen below the pre-arranged value for each step. The operator cannot manipulate or alter this adjustment.

In effect, this means the operator can start and accelerate the hoist only at a fixed maximum of permissible power, although he can retard and cut off power as quickly as he wishes, thus providing against abusive handling of the hoist and motor, and limits the power peak drawn at starting.

Extra resistances may be provided when considered necessary for special slow speed for shaft inspection, etc. These must have ample heat-radiating capacity on the various steps to dissipate the heat of control involved in slow speed running.

All resistances must be arranged in a suitable location, and are installed, with all contactors and automatic relays, in a small fireproof room adjacent to the hoist.

For motor capacities of 400 h.p. or greater "liquid control resistance" would be used in preference to metal resistance, as giving a more even control current gradient.

An Emergency Enclosed Switch is provided on the operator's platform which can be opened to sever the control circuit, thus cutting off power to the motor and automatically applying the brakes in case of any derangement in the control.

A High-Torque Push-Button Switch, which can be held pressed in by the operator's foot, will permit the closing of two or three contactors and cutting out the first steps of resistance in the rotor circuit, to provide extra starting torque by the motor, irrespective of the predetermined maximum current controlled by the current limit relays. The object of this device is to enable the hoist operator in emergency to lift extra-heavy loads, which he may be called upon to do in case of accident, when at the same time the hoist may be operating on single drum through one rope drum being out of gear or disabled. This maximum torque button can function only while it is held pressed by the operator, and its location is so arranged as to discourage its use, save in emergency.

All automatic devices are of the closed circuit type, thus providing, should any control wires break or be disconnected, that the motor stops and the brakes are automatically applied by gravity. Should the electric power supply fail or cut-off from overload or other cause, the hoist stops ; should power come on again and the operator has neglected to put his controller to the "off" position, the motor will *not* start till after the handle is at the "off" or neutral position ; then he must start in the ordinary sequence again.

Overwind protection is provided by a track switch for each compartment of the shaft, arranged at a suitable distance below the head gear. This switch is opened by the cage shoe passing, and actuates the control circuit, the hoist power is cut off, and the emergency brake automatically applied. After overwinding, the operator cannot start the hoist or motor before operating a "backing out" switching device ; this will permit the cage to back out only, no matter which compartment has overwound, thus preventing trouble from further operation in the wrong direction, which may happen should the operator get "flurried"—most likely to occur immediately after an overwind. When desirable, an overwind device can be geared to operate from the hoist drums to act similarly to the above "track switch."

An over-speed device is not considered necessary, as the induction motor has the inherent property of not running above its designed speed—in these installations, 400 r.p.m.

Where found advisable, a centrifugal governor, suitably geared to the hoist drums, actuates the motor control, and will compel the operator to slow down near the end of the travel, after which the overwind gear functions as previously described.

Disabled Operator's Stop, for which special provision is made to automatically and instantly cut off power, apply the brake, and stop the hoist, in the event of the operator becoming disabled from any cause or leaving his control handle for an instant (known as "dead man's stop.") This provision is incorporated in the master-controller, and is always ready and effective without the special exercise of any physical or mental effort on the part of the operator (see Fig. 14).

The Master-Controller is small, and operates with very little exertion by the "driver." This is appreciated after a heavy operating shift. The current carried is very small, and the wear of parts is negligible. The control lever is about 10 in. long, and moves in a vertical arc. It has a forward movement for clockwise or forward rotation, and backward movement of the handle for counter-clockwise or backward action of the hoist power. A mechanical latch is provided for retaining the handle in the centre or neutral position when idle ; this latch has to be depressed by the operator's "hand palm" when he wishes to move the controller or hoist in either direction. This first action of clearing the holding latch before starting also electrically raises the solenoid brake and permits the first contactor step to close and supply power to the hoist motor in the proper direction and order under control of the automatic current limiting relays, as previously explained. Nothing can operate without this first preliminary movement of the latch, and neglect to retain, or by relaxation of pressure to hold off the latch of the operating lever for an instant, will cause the hoist to stop, and can then only restart in proper sequence again.

From this it is seen that the whole electric control is very effective and extremely simple—practically one small movable lever, requiring no special attention or exertion from the operator and giving automatic protection for all likely conditions.

The driver or operator is provided, on his platform, with two ammeters in clear, open view. One meter indicates the current being used by this motor at any time whilst running, and the other meter indicates the total current being used at any time by the combined electric hoists on the system. The first meter is a guide as to the duty of each hoist. The second meter indicates the full loads combined, and enables each individual hoist driver to refrain from "starting" simultaneously, thus reducing heavy peak loads, which is of considerable advantage with a moderate-sized power station. This delayed starting only amounts to 2 or 3 seconds, but is effective, although not required now at the South mine under normal generating conditions when the large 3750 k.v.a. turbo-generator sets are in service.

South mine "*Chairs*" *Indicating System* is fitted to the hoist cage indicators for indicating the position of "chairs" in the shaft, as described in Paper No. 172, by W. E. Wainwright, Transactions, xiii., A.I. of M.E., 1909.

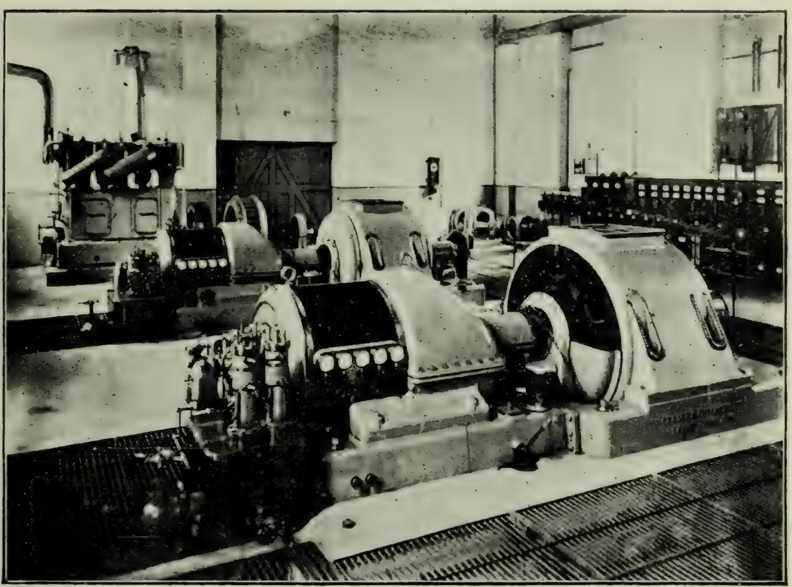


FIG. 1.
South Mine Main Power Plant Interior.

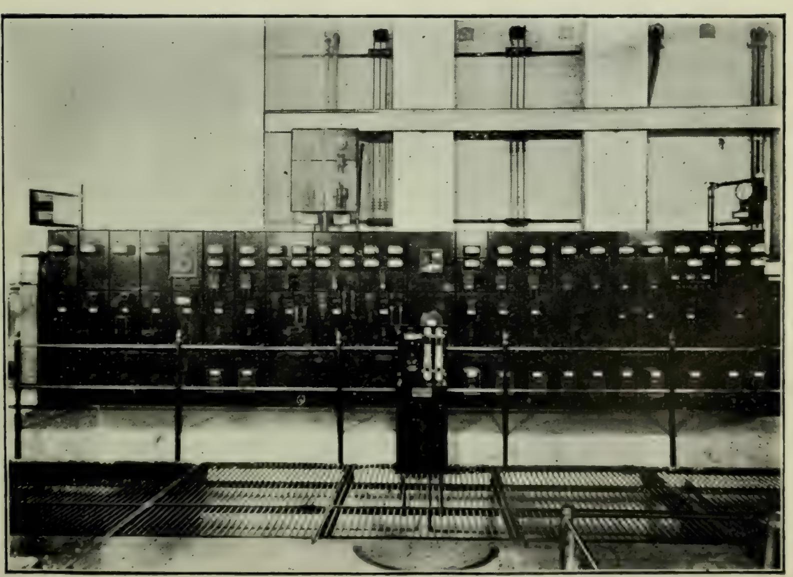


FIG. 2.
Front view of Main Switch Board in Power House.

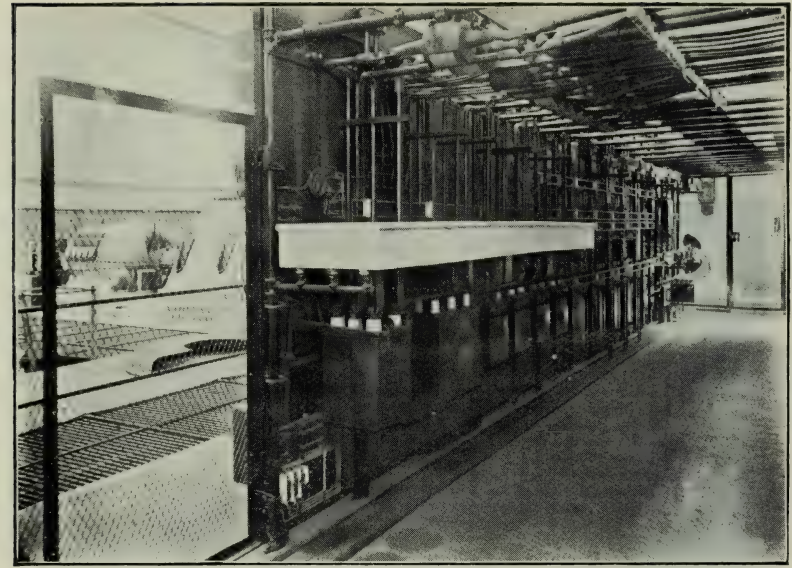


FIG. 1.
View behind Main Switch Board in Power House.

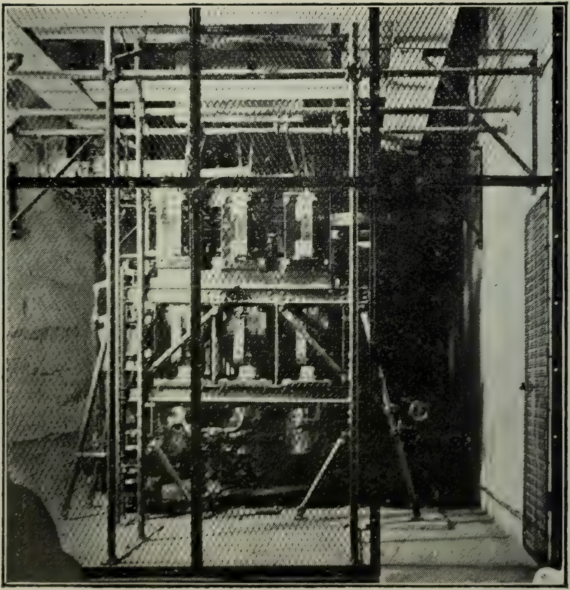


FIG. 2.
End view of Main Switch Compartment.

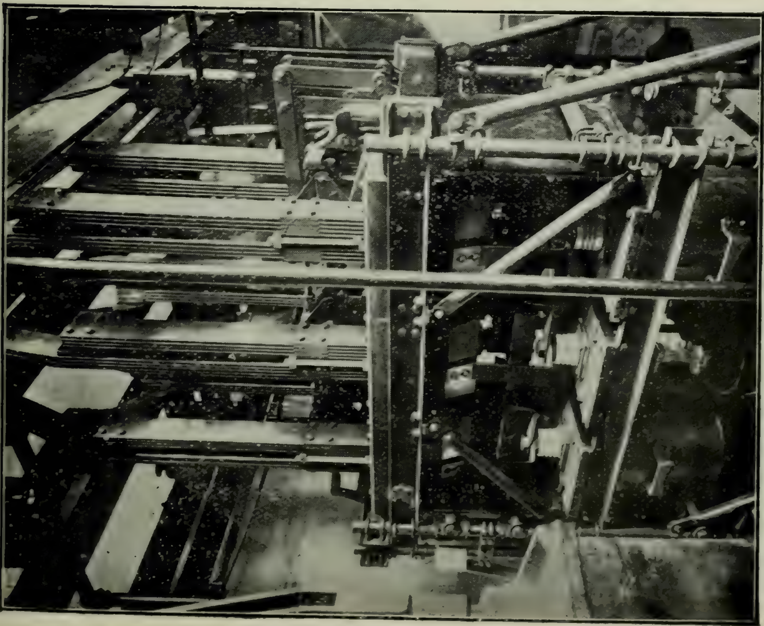


Fig. 1.

5000 Amp. Automatic Oil Switch for 3-phase Turbo Generator.

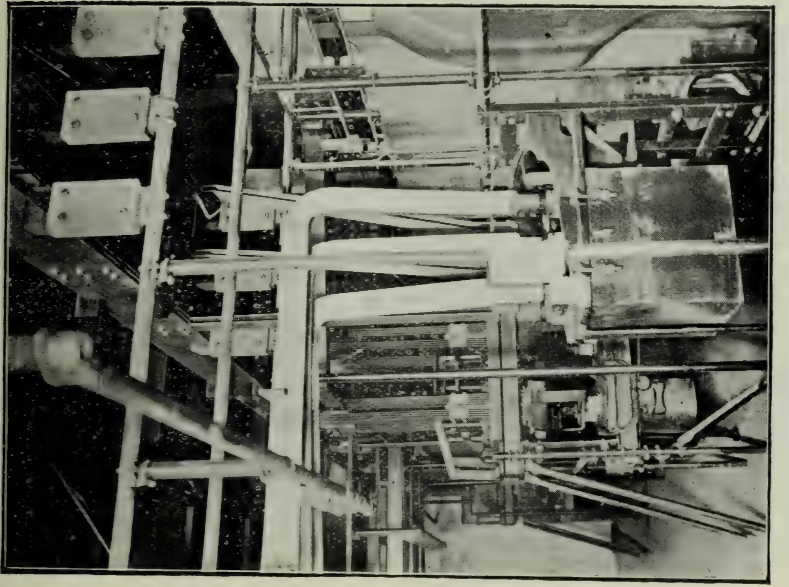


Fig. 2.

One of the 2000 Amp. Main Feeder Switches also one of the 5000 Amp. Generator Switches.

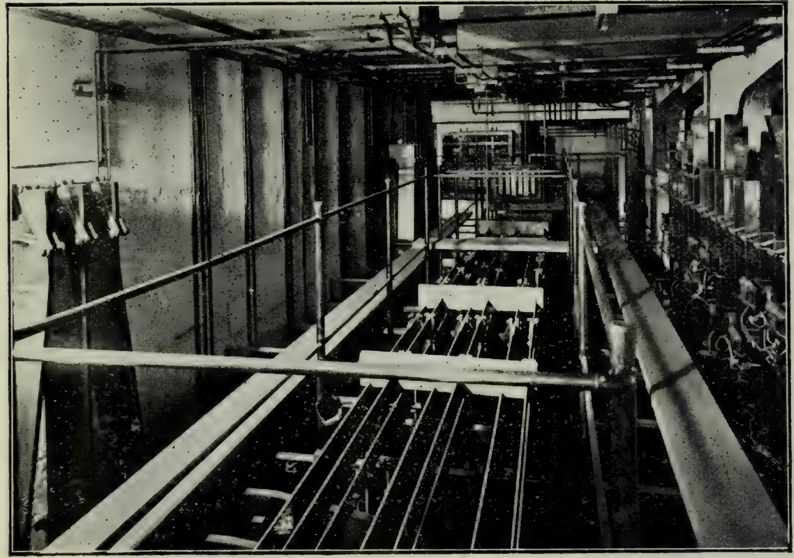


FIG. 1.
View taken overhead of Busses and Switch Gear.

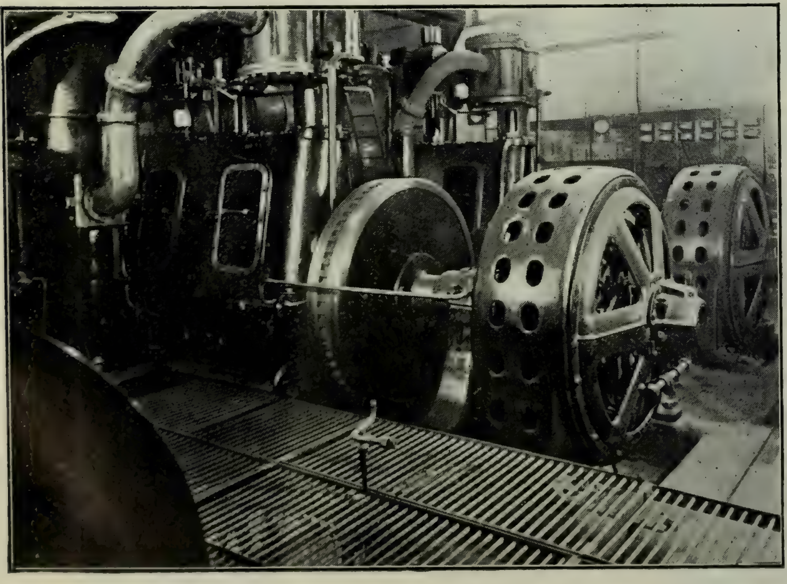


FIG. 2.
Two 440-H.P. Induction Motors.

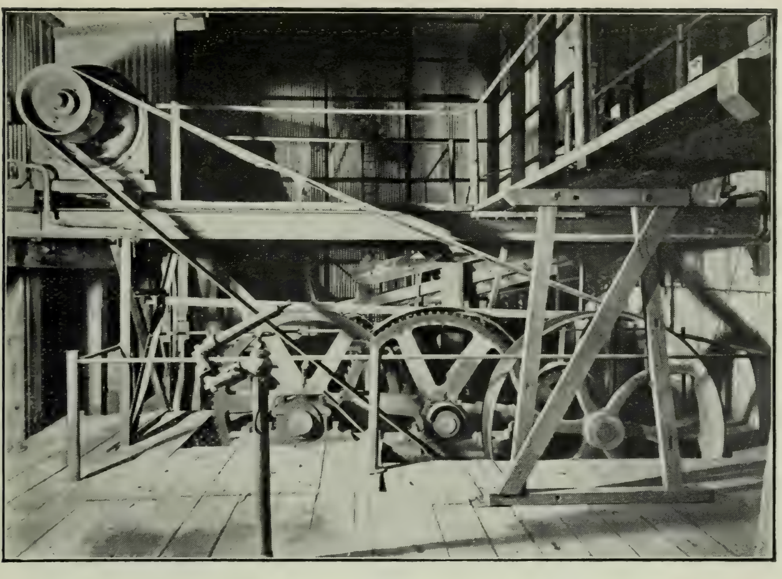


FIG. 1.

75-H P. Motor belted and geared to 18-in. x 36-in. Crushing Rolls.

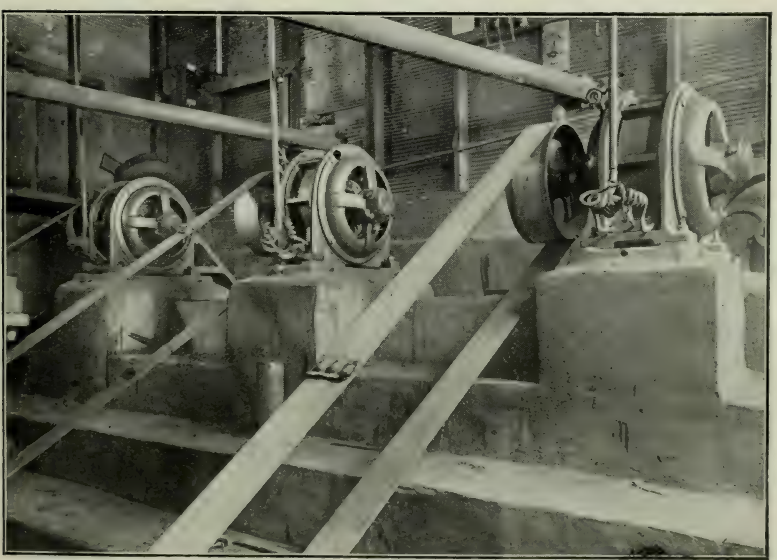


FIG. 2.

Three 15-H.P. 800 R.P.M. "Squirrel-Cage" Motors.

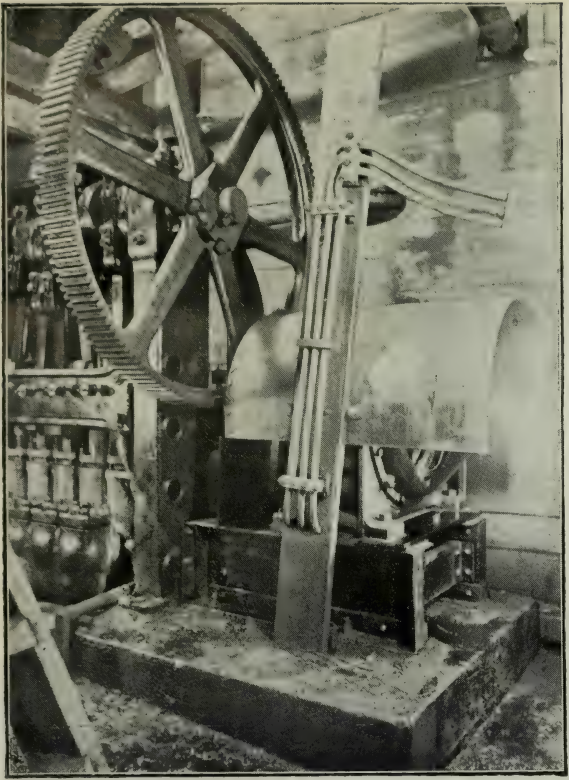


FIG. 1.

75-H.P. Geared Electric Mine Pump.

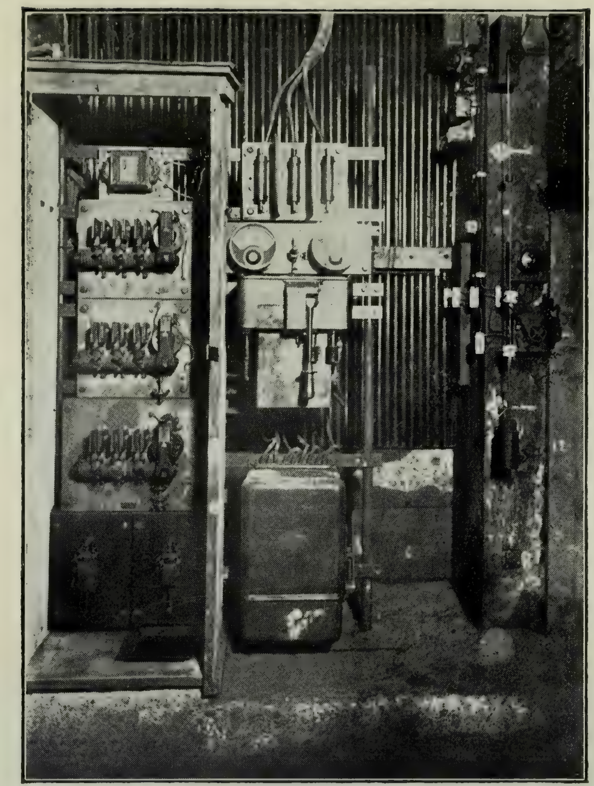


FIG. 2.

Control Gear for automatically starting and stopping 75-H.P. Electric Mine Pump.

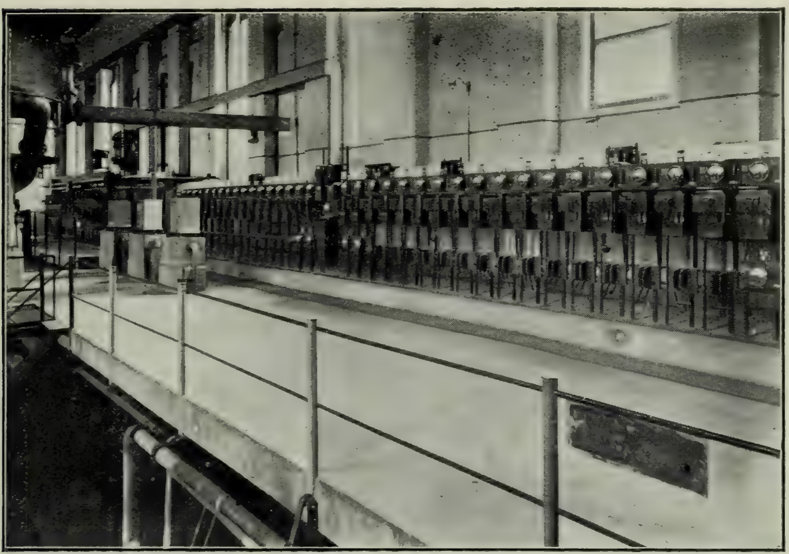


FIG. 1.
Grouped Remote-Control Switchboard for Motors

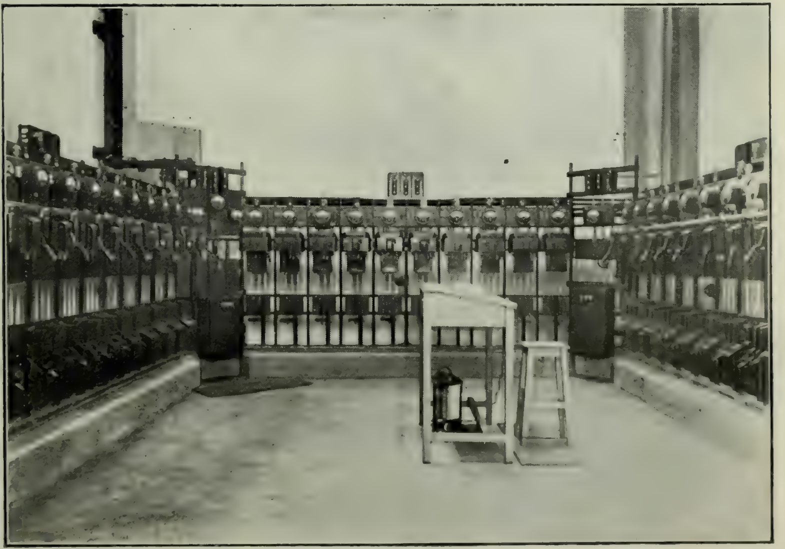
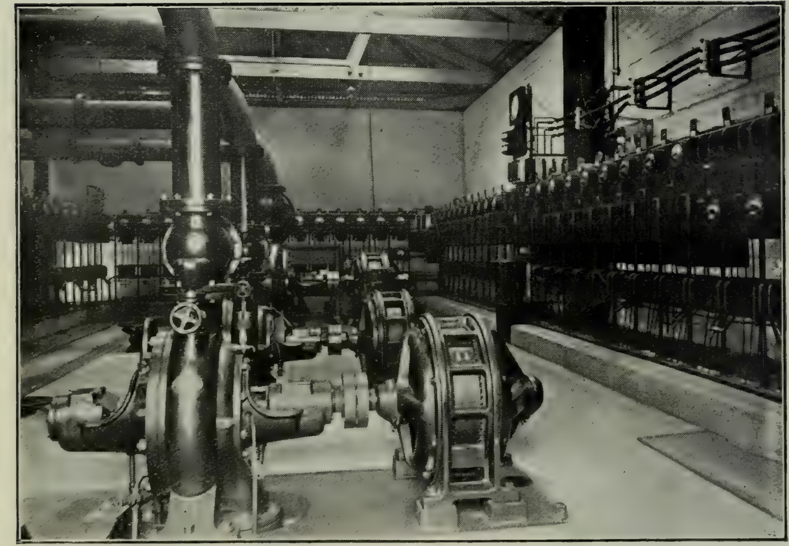


FIG. 2.
No. 1 Grouped Motor-Control Depot.



No. 2 Motor-Control Station for 80 Motors sizes 5 to 250 H.P. also showing 4 Motors 30-50 H.P. direct-connected to 4 Centrifugal Mill Water Pumps.

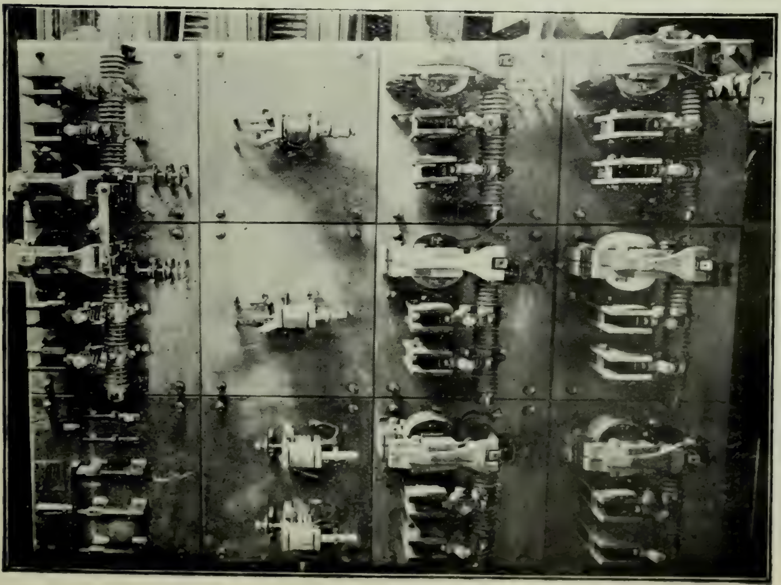


Fig. 1.

Magnetic "Contractor" Control Panel.

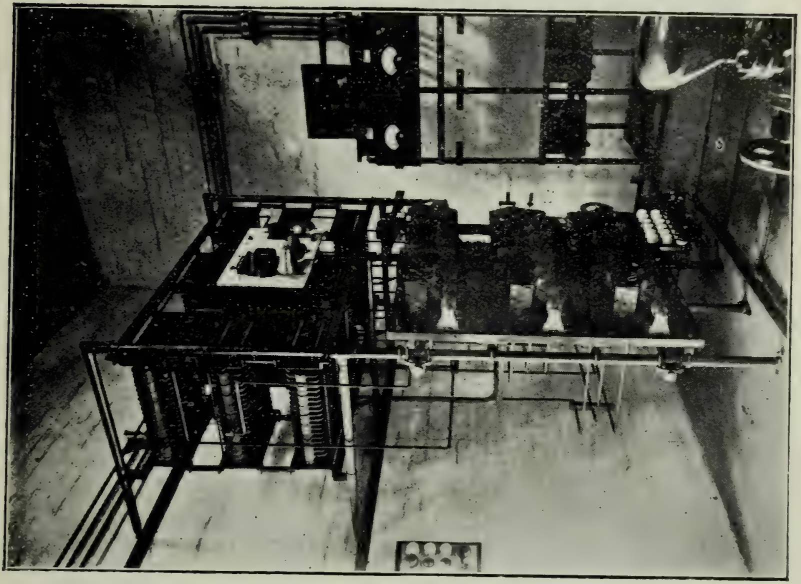


Fig. 2.

Control "Contractors" and Rotor Resistance.

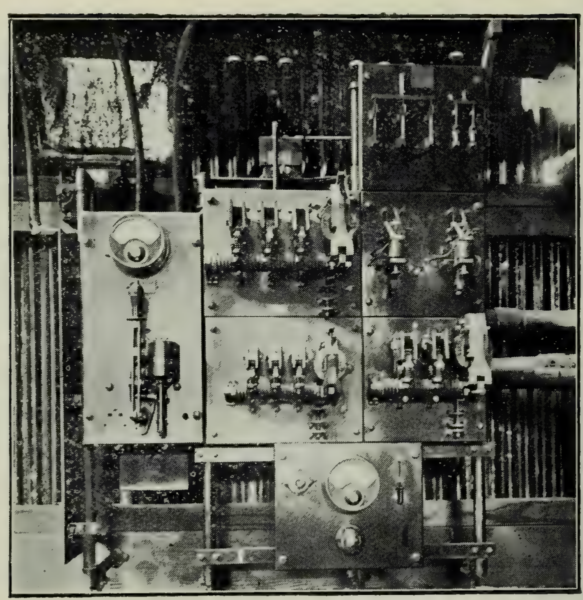


FIG. 1.
Control "Contractors" for one Panel of a bank of eight 200-H.P. Slip-Ring Motors.

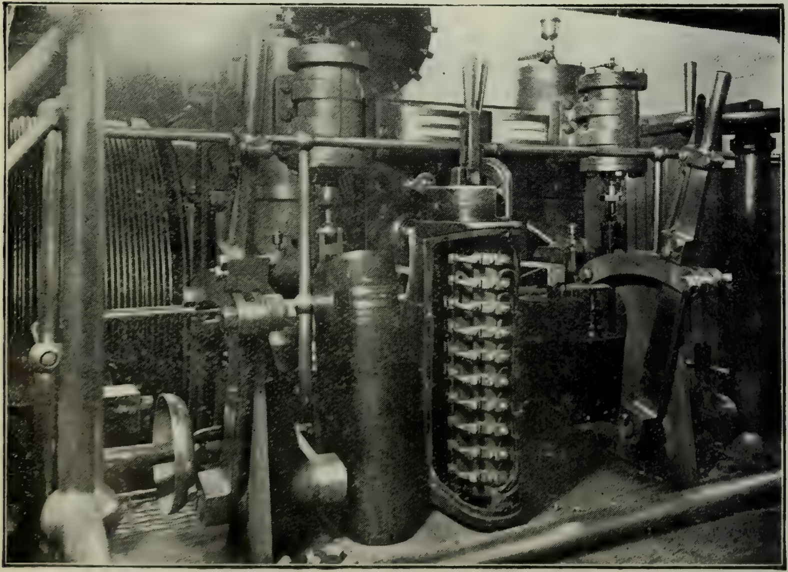
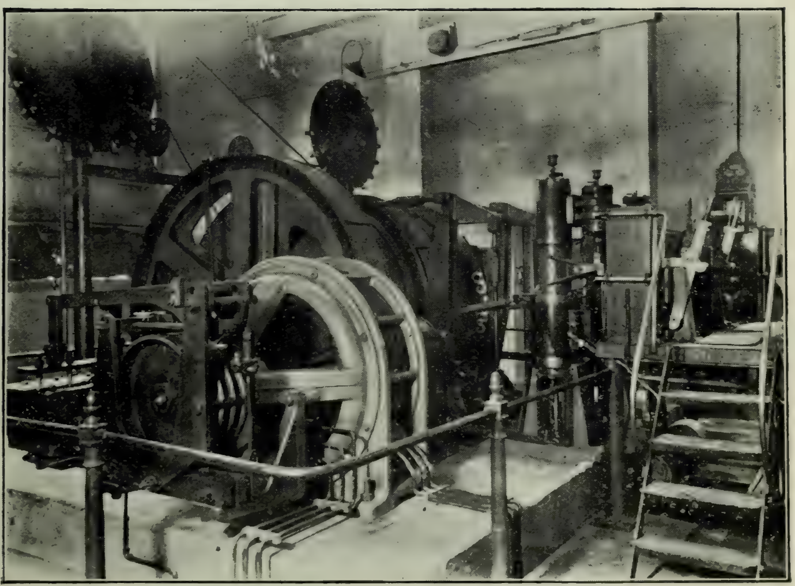
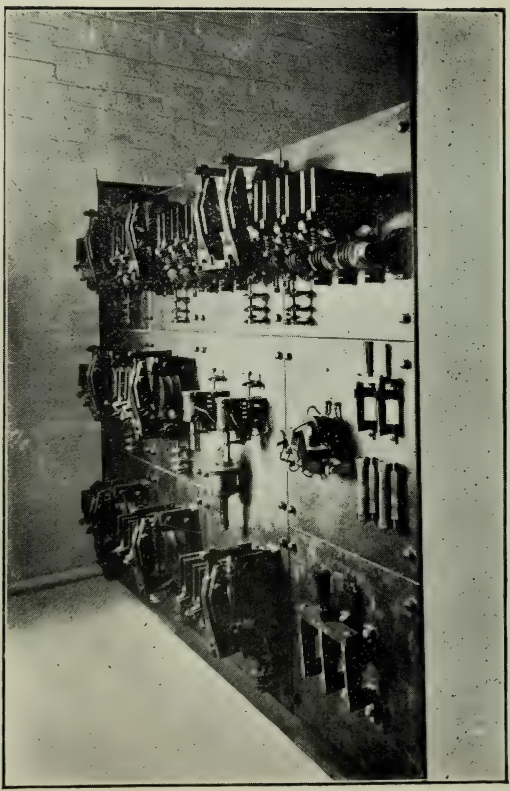


FIG. 2.
Hoist Operators Control Platform.



300-H.P. Geared Electric Mine Hoist with Solenoid Emergency Brake. Electric-Magnetic Control.



Mine Electric Hoist.

"Contractor" Control Switchboard, located in small fireproof room near hoist, showing automatic equipment for 300-H.P. Induction Motor.

BURMA CORPORATION LTD.: A DESCRIPTION AND REVIEW OF EARLY CHINESE AND PRESENT-DAY OPERATIONS.

BY JOHN W. MOULE.

THE Corporation owns a large silver–lead–zinc mine and treatment plant in Upper Burma. The Chinese, as far back as 1412 A.D., mined and smelted the ore for its silver contents, and this paper has been prepared with the belief that a review of past and present-day operations will be of interest to members of the Institute.

Many figures and statements regarding the Corporation's activities have already been published, so the writer has felt at liberty to incorporate in this article some of the estimates prepared by him on the field during 1920.

The records of the Geological Survey of India for 1909, now out of print, contain an excellent article by Messrs. La Touche and Brown, aided by Dr. Malcolm Maclaren, on the early Chinese history and also geological notes on the Bawdwin mine ; and, as these records are authentic, liberal use has been made of the information and illustrations given in the above records. A later and more comprehensive paper on the geology of the district, by L. Coggin Brown, was published in Records Geological Survey of India, 1918, Vol. 48, p. 165.

The Bawdwin mine is situated in one of the Northern Shan States. These States lie to the north-east of Burma proper, and between it and the Chinese border of Yunnan. The Shans are a distinct race, with a language and customs of their own. Each State has its own "Sawbaw," or ruler, who in any important business acts under the guidance of a British official responsible to the Government of India and Burma. These Government officials are of a very high standard and of unimpeachable integrity.

The Bawdwin mine is known to the Chinese of Yunnan as "Lao Yin Chang," or the "old silver mines," and is reported to have been first worked in the ninth year of the reign of Yung Lo, of the Ming dynasty, 1412 A.D.

Bawdwin is almost 600 miles from the seaport town of Rangoon, and 200 miles north-east of Mandalay. It is but 40 miles from the Chinese border, and the early Chinese occupation of the mine was permitted by the local inhabitants merely as a commercial exploitation, and had no political significance.

The mine was a source of wealth to the Chinese until the reign of Tung Chih (1860–1875 A.D.), when it was deserted for three main reasons—

(1) The Panthey (Mohammedan Chinese) rebelled against the reigning powers in Yunnan, and made life and property insecure.

(2) The Kachins, a race living in the northern hills, advanced, and ruthlessly raided the mining population. It is possible that at this time the very extensive lines of entrenchments were constructed. They extend for some miles on the crest of the hills surrounding the mine, and can still be seen along the ridges and spurs of the hills. It may also account for the manacled Chinese which were recently discovered in some of the old workings.

(3) Increasing difficulties were experienced in procuring fuel supplies and in extracting the ore as the galleries or workings approached the level of the stream flowing through the valley, for the Chinese only had very primitive bamboo pumps.

The mine was finally abandoned by them about 1868 A.D.

The Burmese kings, Minden Min and Thebaw, later made several attempts to work the mine. The Burmese were, however, not so skilled in mining and metallurgy as the Chinese, and their efforts resulted in failure.

EARLY CHINESE METHODS.

The following notes and illustrations*, mainly gathered from the geological records, enable one to form a fairly accurate idea of the Chinese mining and metallurgical methods.

The sides of the gorge afforded a ready means of attacking the ore-bodies by driving horizontal or slightly inclined adits.

Some of the excavations were driven almost vertically upwards, either to afford ventilation or to follow the richer ore, and were ascended by steps cut in the rock; others descended below the level of the stream.

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* The Institute is indebted to the Geological Survey of India for permission to reproduce these illustrations.

Very little timber was used, and in most cases it was not necessary, on account of the manner in which the "backs" or roof of the galleries were arched. These galleries varied in height from about 3 ft. to 5 or 6 ft., and were usually well proportioned. At times they expanded into irregular cavities, where, no doubt, large bodies of ore had been encountered, and in places some of these old workings had been timbered and filled.

There does not appear to have been any definite system of levels and headings, but on the whole one must, all things considered, concede the excellence of the Chinese methods, for the European company which first exploited this property collected and treated 150,000 tons of lead slag, containing about 40 % lead and under 1 oz. of silver per ton, which the Chinese had produced from the ore they had mined and treated.

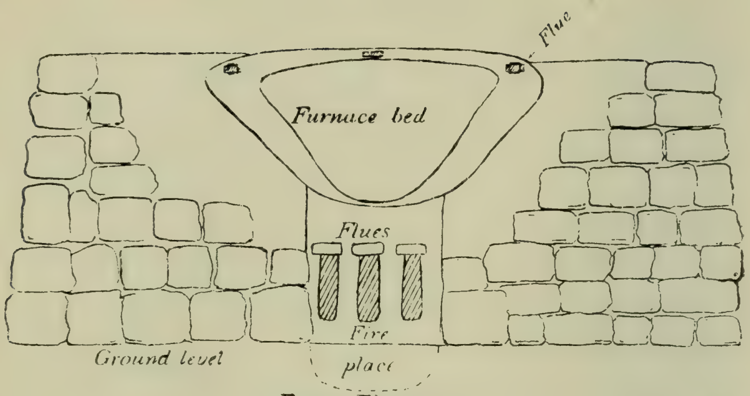
The ore was in all probability a mixture of lead sulphide, sulphate, and zinc-blende, and, when mixed with gangue, was powdered in mortars of about 12 in. in diameter. One of these mortars is illustrated in the foreground of Plate I., fig. 1, and is hollowed out of a solid piece of sandstone or grit.

The crushed ore was then heaped upon a level piece of ground close to the river. A tank about 8 ft. square and 2 ft. deep, for holding the water, was built near the floor. The powdered ore was then gradually washed down a chute paved with stones, sloping at an angle of about 35°, the lighter material being washed away, while the heavier silver-bearing ore was left behind.

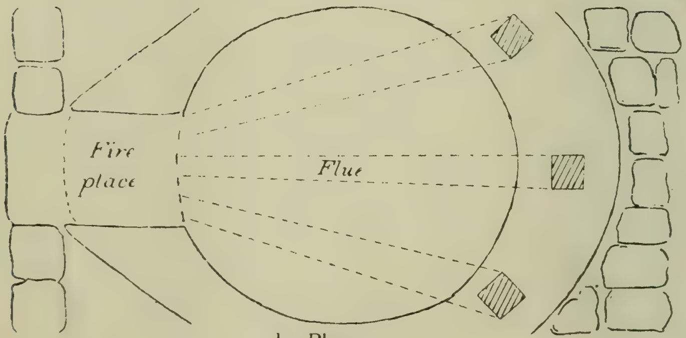
These floors were also used in separating ore from the talus of sandstone fragments filling the bottom of the ravine, and extended down the valley for nearly a mile below the mine.

The ore, after receiving its preliminary treatment, was then possibly subjected to a preliminary roasting before being mixed with charcoal and smelted in the large furnaces (see illustrations, Plate I., figs. 1 and 2). These were placed on convenient positions on the hillsides and above the bed of the stream, so as to allow sufficient room for dumping the slag.

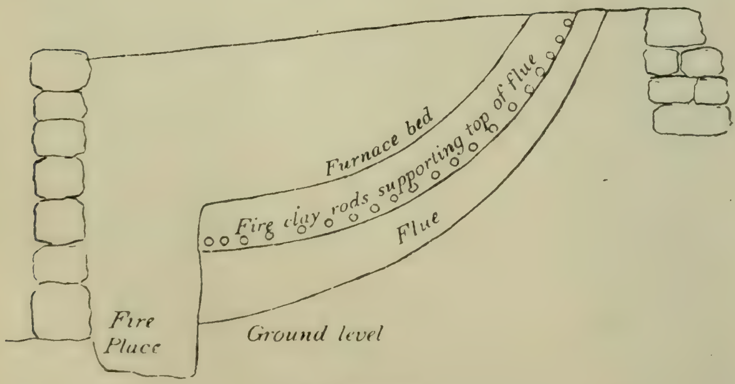
The furnaces were of simple construction. Each consisted of a concave fire-clay bed or hearth of circular or oblong shape, and sloped from back to front, where it was narrowed, imbedded in a mass of hard clay built up inside a wall of stone or large bricks. The hearth



a. Front Elevation



b. Plan



c. Section

FIG. 1.

LARGE SMELTING FURNACE.

Scale—1 in. = 4 ft.

From Records Geol. Sur. of India.

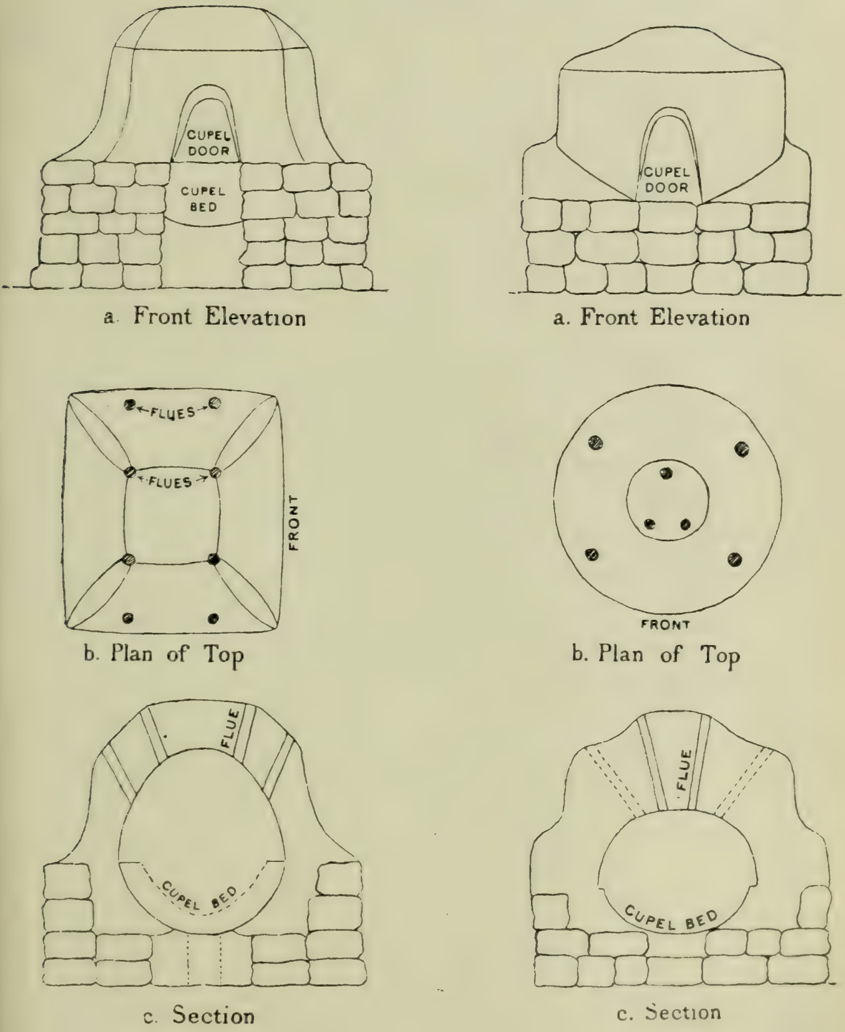


FIG. 2.

SQUARE CUPEL FURNACE.
Scale—1 in. = 4 ft.
From Records Geol. Sur. of India.

ROUND CUPEL FURNACE.
Scale—1 in. = 2 ft.
From Records Geol. Sur. of India.

was heated from beneath by three flues, and the roof was supported by rods of fire-clay about 1½ in. in diameter and radiating from the back of the fire-box.

As the reduction proceeded the enriched lead ran down to the front of the hearth, percolated through the mass of fuel and ore filling the front of the furnace, and collected in a hollow beneath the flue.

The resulting lead was cupelled in beehive-shaped furnaces (see Fig. 2), several of which were left by the Chinese in a very good state of preservation. The measurements were taken from a row of twelve built against the hillside near the bank of the stream. These round furnaces appear to have been used for a preliminary extraction of the silver, the final purification being effected in the square furnaces (see Plate I., figs. 1 and 2). Two of which were found, and evidently were considered to be of more importance than the others, as they were enclosed in a solid stone building, which had probably been roofed over. Outside the door of this was a stone table, on which the silver may have been weighed and packed for despatch to China. The silver was probably worked to a fineness of about 975, which is the fineness of the silver produced by the natives in the Bawdwin district.

In addition to the smelting furnaces, the remains were found of two or three small furnaces of the ordinary native type (Fig. 3). These were apparently used for assay purposes, as close to one of them was found a small sandstone mortar, about 3 in. in diameter, which could only have been used for crushing very small quantities of ore.

For several miles around Bawdwin the hills are almost entirely denuded of trees, but are thickly covered with grass. This denudation of forests is an invariable accompaniment of Chinese occupation, whether mining and smelting is carried on or not.

The Chinese were evidently very able prospectors as well as miners, for there are numerous, though small, heaps of slag, from both lead and copper ores, scattered throughout the Northern Shan States; but so far nothing has been found in them to equal in size those formerly at Bawdwin, nor have any deposits been found which have not been already worked by the Chinese.

It is, however, reported that large slag heaps exist in the Wa States, and near the Chinese border (see Fig. 4). These States, however, are inhabited by head-hunters, and are at present unadministered. The

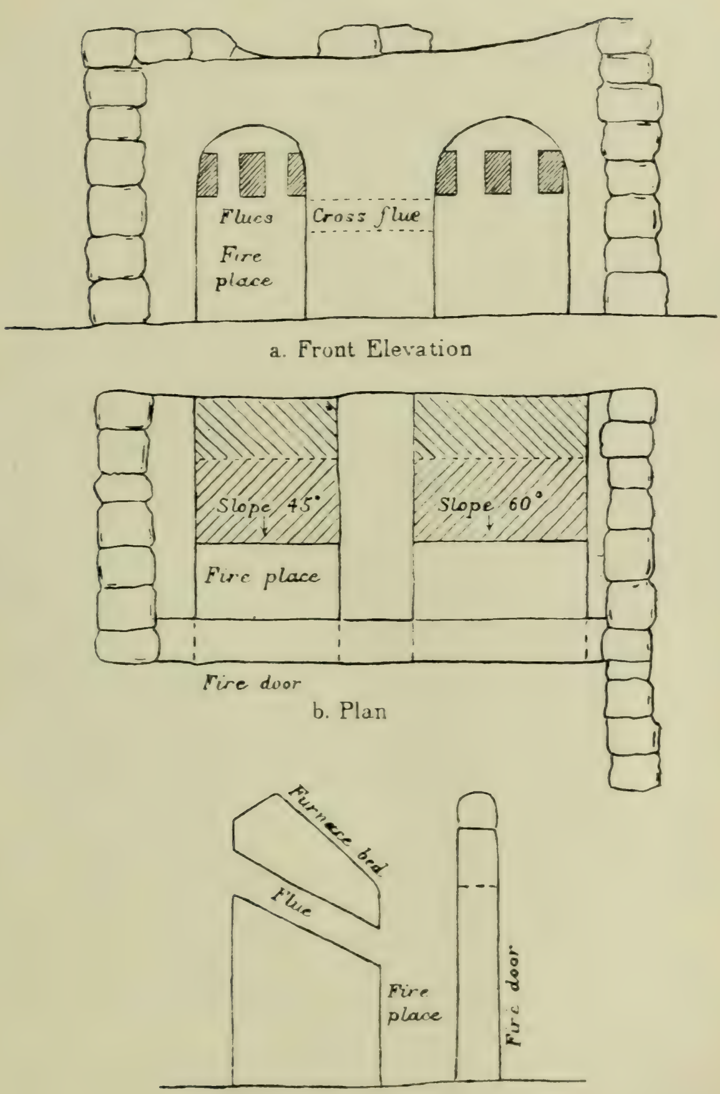


FIG. 3.

SMALL SMELTING FURNACE.

Scale—1 in. = 2 ft.

From Records Geol. Sur. of India.

Government will not permit any British subject to enter these States until they are subdued, so that it will be quite some years before the costly railway, which will have to pass over difficult country, can be built and tonnage obtained from these deposits—that is, if they prove to be as extensive and as remunerative as is anticipated.

BAWDWIN MINE.

The mine is about 8 miles N.N.W. from the settlement of Namtu, which is situated on the Namtu River.

GEOLOGY.

The general sequence of rocks to the southwards of Bawdwin, so far as has been ascertained, is as follows :—At a variable distance of from two to five or six miles to the west of the Namtu River, which, in this part of its course, runs almost due north and south through a deep gorge, is found an ancient land-surface, composed of slates, quartzites, and grits of pre-Ordovician age, along the eastern border of which rocks of Ordovician, Silurian, and Devonian age were deposited in succession.

These rocks have been traced northwards from the neighbourhood of Hsipaw, forming a broad belt along the valley of the Namtu. Along the outer or western edge of this belt Silurian grits and sandstones (Namshim beds) are found resting directly upon the upturned edges of the rocks forming the old land-surface, with a very irregular boundary, deeply eroded by the tributaries of the Namtu River.

Near the base the Silurian beds consist of coarse felspathic grits, frequently pebbly, which are succeeded by finer sandstones as followed from the old shore-line, and finally by fine-grained sandy marls. They overlap the older Palæozoic or Ordovician beds in such a manner that the later are only visible at a considerable distance from the shore-line, where they happen to have been brought to the surface by subsequent earth-movements.

The Ordovician beds consist of shale, marls, and limestones, usually rich in characteristic fossils, and are known locally as the Naungkangyi beds. At the top of them, immediately beneath the Namshim sandstones, is found a thin but persistent band of shales—the Panghsapyi graptolite band—containing graptolites of Llandovery types resting

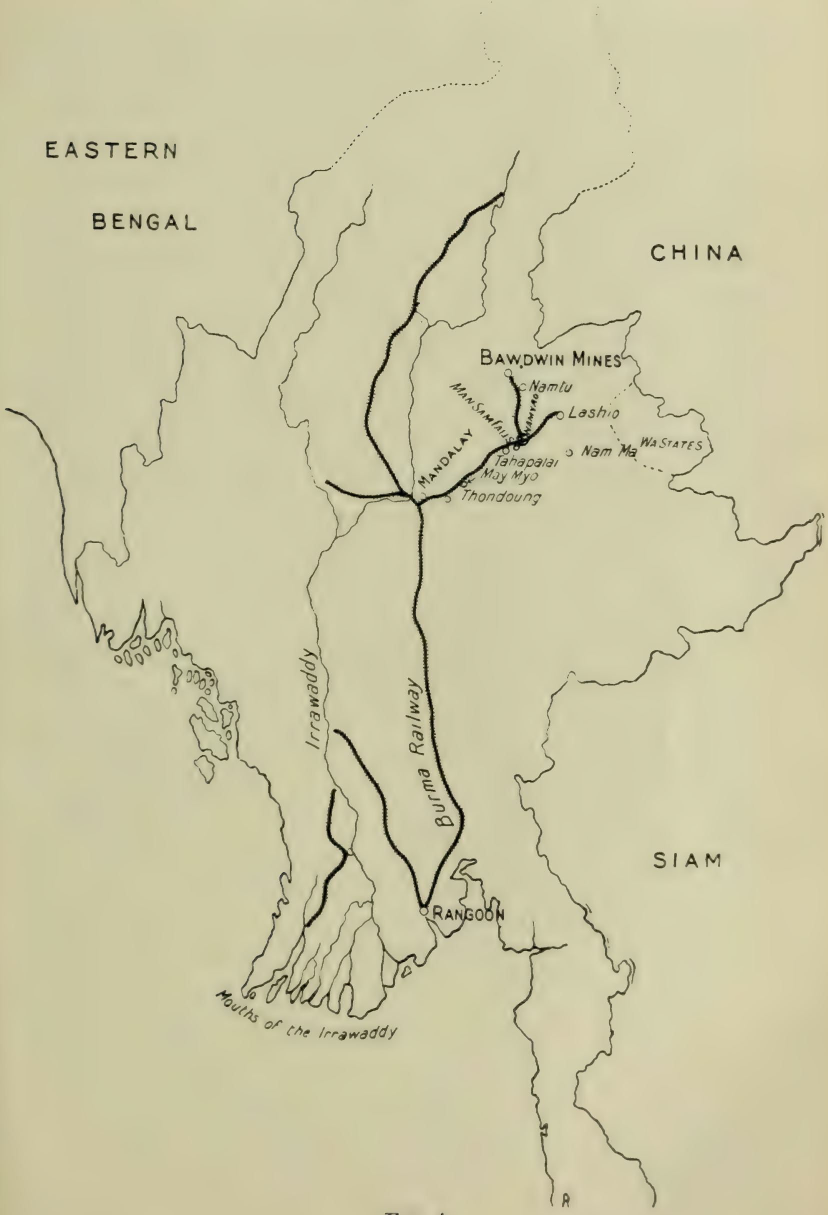


FIG. 4.
BURMA.
Scale—1 in. = 160 miles.

upon a band of purple calcareous shales, which latter represent the purple Hwe Mawng beds of the ranges between the plateau and the

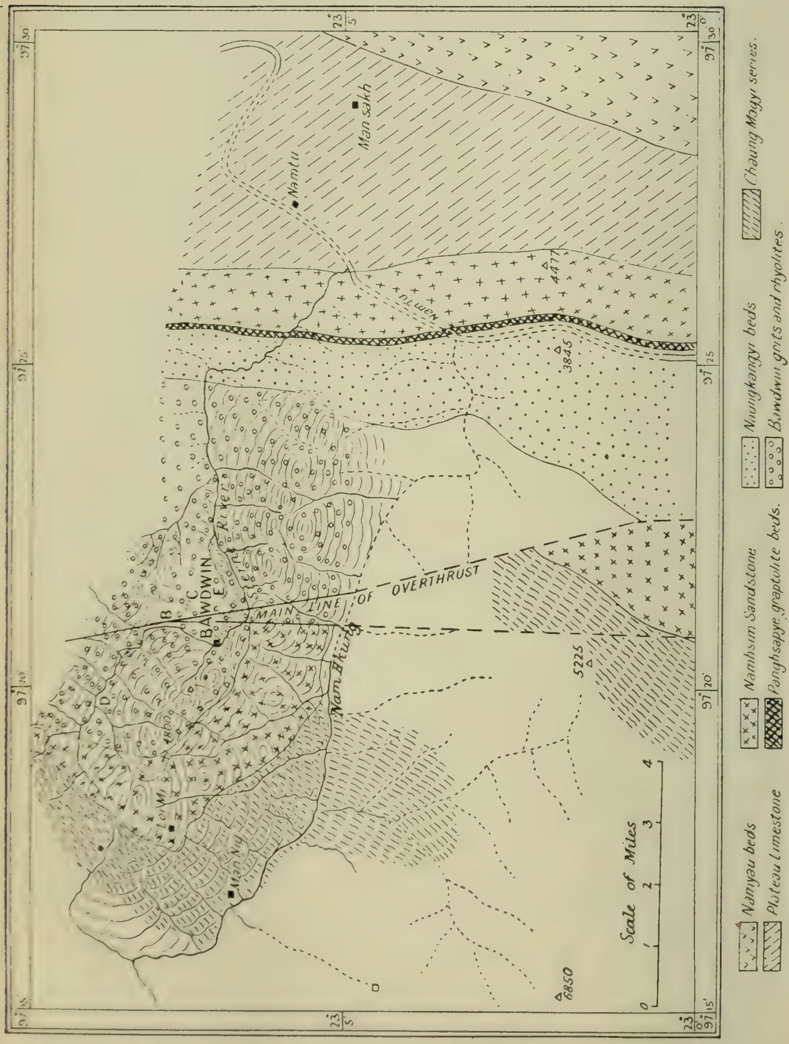


Fig. 5.

Sketch Map of the Bawdwin District.

From Records Geol. Sur. of India.

Salween River, to the south of Lashio. The Namshim sandstones are succeeded above by the limestones of the plateau, of Devonian or Carboniferous age, which does not extend to the west of the Namtu River in the southern part of the area, but crosses it at the mouth of the Sterne River, a tributary to the Namtu (see Fig. 5).

The chief structural feature in this belt of rocks is a great overthrust, which has been traced almost continuously to Bawdwin from a distance of 30 miles to the south, and running parallel to the Namtu River for the greater part of its course. Along this overthrust the Naungkangyi beds have been pushed up westwards over the Namshim sandstones, so that a broad belt of the former intervenes between two areas occupied by the sandstones, the latter to the east being found in their normal position under the plateau limestone, and forming a precipitous scarp along the left bank of the river, with the graptolite and purple bands at its base, while to the west the sandstones dip beneath the Naungkangyi beds along the plane of the fault (see Fig. 6). As will be seen the fault appears to take a north-north-westerly trend, and, leaving the band of Naungkangyi shales, traverses the floor of older rocks underlying the Naungkangyis in the neighbourhood of Bawdwin.

These older rocks consist here mainly of coarse felspathic grits and rhyolitic tuffs with thin flows of true rhyolite, succeeded above by finer-grained reddish-brown sandstones, not easily distinguishable from the Namshim sandstones to the south, but without fossils.

The sandstones are found extending for four or five miles along the gorge of the Sterne River below Bawdwin, with a regular easterly dip, until they disappear beneath the Naungkangyi shales. The overthrust passes through the felspathic grits at Bawdwin itself, and it would appear the mineralization of the rocks of that area is due to the intense crushing and dislocation of the rocks caused by it.

A minor line of disturbance of the same nature as the principal overthrust appears to branch off from the latter at Bawdwin, taking a more southerly course, where it later dies out. The most intense amount of dislocation and crushing has therefore taken place at Bawdwin, where these two lines unite.

Over a broad zone parallel with the plane of overthrust, through which the Sterne River has excavated a deep, narrow gorge, the crushing has been of the most intense description, and shear planes and fissures thus produced have afforded an easy passage for the mineralizing agencies from below (see Fig. 6).

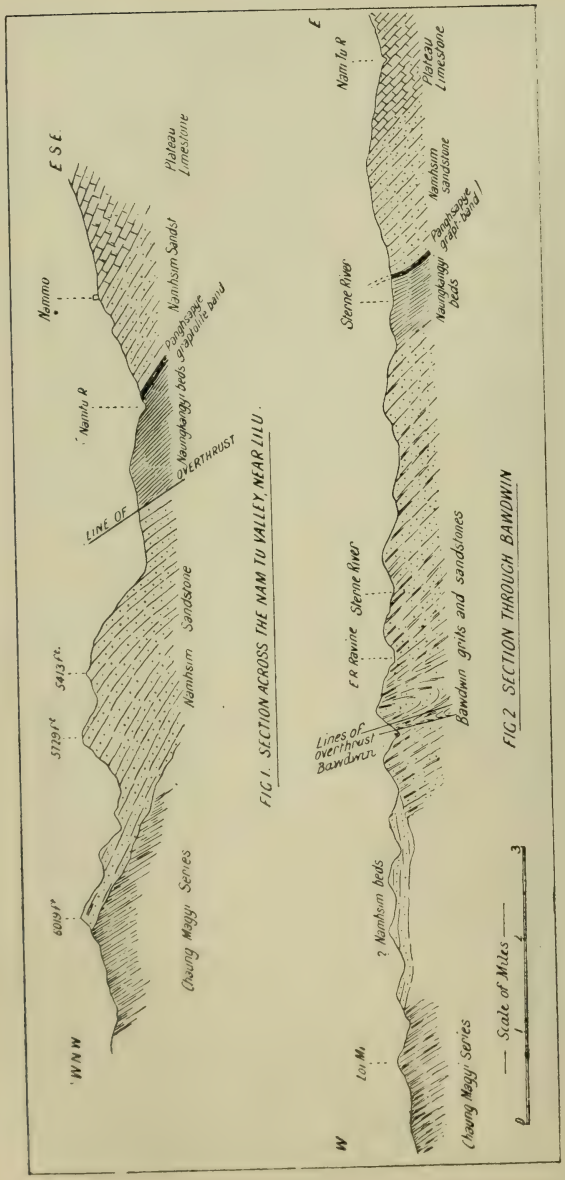


FIG 1. SECTION ACROSS THE NAM TU VALLEY, NEAR LILU.

FIG 2 SECTION THROUGH BAWDWIN

Fig. 6.

Section through Bawdwin.

From Records Geol. Sur. of India.

In the early days of its exploitation under European supervision a considerable amount of unprofitable work was done in the mine, as the faulting of the main ore-body had not then been realized, and it is only since 1913 that successful and active development of the Bawdwin deposit has been carried out, and from which time many thousands of feet of driving, cross-cutting, and rising have been accomplished.

The ore-body was first attacked by sinking a shaft at Bawdwin in the Sterne River valley, but later, when it was known that a large body of ore really existed, and the difficulties of shaft-sinking and consequent heavy pumping had been better realized, a bolder scheme of opening up the ore-body at a lower level by driving a tunnel from Tiger Camp, was decided upon. This tunnel is 9 ft. by 8 ft. inside measurement, and is 7400 ft. long, and 653 ft. below the zero level at Bawdwin. It corresponds to No. 6 level, and through it all the water is now drained and ore hauled to the mine ore-storage bins at Tiger Camp (see Plate II., fig. 1). Considerable difficulties were encountered in driving this tunnel, such as bursts of water and running ground, and as these troubles reappeared each wet season and caused a temporary blockage of the tunnel through the timbers collapsing, it was found necessary to begin replacing the timber supports, where the ground is at all treacherous, with a masonry arch.

The mine workings extend north and south over a length of 2500 ft., but at one point the lode has been thrown approximately 600 ft. along a fault known as the Yunnan fault, while at the south end of the ore-body another fault, known as the Hsenwi fault, has intersected and, up to the present, cut off the payable values.

Besides these two main faults there are other and numerous smaller ones, but they have little or no effect on the working of the mine. The northern body is known as the "Chinaman Lode," and at one point on No. 4 level is 140 ft. wide. The southern body is called the "Shan Lode." There are also a couple of narrow lodes or veins, named the "Burman" and "Kachin," close to and approximately parallel with the larger bodies, and all are almost vertical and exist in the rhyolite.

ORE RESERVES.

The ore reserves are officially stated at approximately 3,600,000 tons of silver-lead ore, assaying silver 24 oz., lead 26 %, and zinc

19 % ; and 300,000 tons of copper ore, assaying silver 23 oz., lead 13 %, zinc, 8 %, and copper 11 %. This copper ore will have to be won by selective mining, for it occurs along the western wall toward the northern end of the main lode, of which it is a part.

In addition to the ore mentioned above, it is estimated that there exists about 3,000,000 tons of low-grade ore, but as this only contains approximately 5 oz. silver and 5 % lead, it is unlikely that the present corporation will ever derive any profit from it.

In these official estimates of ore reserves it has been assumed that mining operations will result in the whole of the ore being extracted, no allowance being made for any losses.

The Chinaman lode has been completely, and the Shan lode almost completely, blocked out above the Tiger tunnel, No. 6 level.

Further development work below No. 6 level, to give additional geological data regarding the ore-body, and to prove its extent and value in depth, is therefore of the utmost importance, for the area of ore exposed on No. 6 level is only 55 % of the area exposed on No. 4 level.

The question as to whether the walls will continue to converge or whether the size of the ore-body will be maintained at greater depth is therefore full of interest, and considerably affects the future of the Corporation.

It is probable that some further additions to the ore reserves may be obtained by (*a*) exploring the parallel smaller lodes or veins referred to above ; or (*b*) exploring the country north and south of the main ore-bodies. The country south of the Hsenwi fault probably offers the better possibilities, as development work to the north of the Chinaman lode has so far been disappointing, and has not added to the ore reserves. On No. 6 level, at 1000 ft. north of the vertical shaft, a cross-cut encountered 16 ft. of ore assaying 15 % lead and 2 % zinc ; but, after raising everybody's hopes, further work proved it to be only an isolated pocket or bunch of ore.

MINING.

The silver–lead ore consists of an intimate mixture of galena and zinc-blende associated with small quantities of iron and copper pyrites and containing about 31 % of insoluble matter, whilst the so-called copper ore is part of the main lode, and occurs along the western wall of its northern extension.

The value of the ore-body is not homogeneous. As a general rule much higher lead and lower zinc values are encountered along the western side of the lode. On the eastern side there are lower lead and higher zinc values, whilst on No. 1 level there is some good-grade zinc ore, and several thousands of tons were, in the early days, shipped as such to Europe.

The ore in the upper levels is gradually oxidizing, and, besides being less amenable to concentration, it generates an appreciable amount of heat, and renders some of the old workings almost unbearable.

Owing to the lack of plant for treating run-of-mine ore, and the necessity for earning profits immediately, effect was given to a policy of selectively mining the highest-grade silver–lead ore low in zinc. The result of this policy has been that mining operations have been practically confined to winning the high-grade lead ore along the western wall, and, where any low-grade ore was encountered, mining operations had to be diverted to fresh places.

The ore is very dense and friable, and requires only a minimum amount of explosives, but very careful timbering was necessary, and, after trying several different methods of mining, the square-set system was finally adopted. Each working face is usually but 3 or 4 sets wide. No precautions have been taken, when opening out on a level, to facilitate future mining operations, when the stope from below has to pick up the old timbers of the sill floor above it, and where the posts stood on the ore. Filling is obtained from rises to the surface, and through the ore-body, whilst all the ore mined is dropped down passes to No. 6 level, where it is hauled in special cars of 4 tons capacity by electric motors to the tipple plant commanding the mine storage ore bins at Tiger Camp (see Plate II., fig. 1).

Haulage and pumping costs are therefore reduced to a minimum, for, as before mentioned, all the water is drained from the mine through this tunnel.

Timbering costs are, however, excessive, for, unfortunately, the quality of the timber available from the near-by tropical jungle is very poor, and renders it necessary to keep a large force constantly replacing the defective pieces. A creosoting plant has recently been installed, and it is hoped that its use will increase the effective life of the timbers in the main travelling ways. Supplies (timber, etc.) are either sent into the mine through Tiger tunnel or from Bawdwin and down the internal shaft, which is distinct from the original vertical shaft.

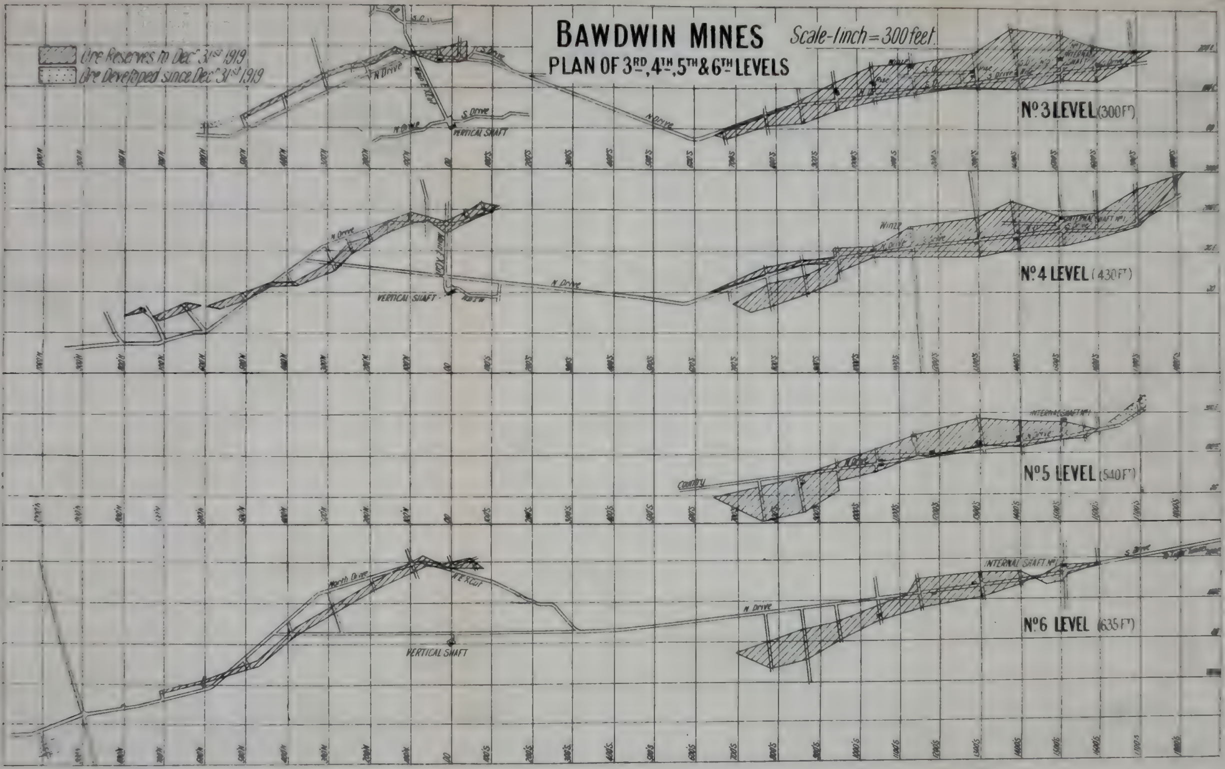
A reference to the mine plan (Fig. 7) will show the position of these two shafts, the relative widths of the ore-bodies on Nos. 3, 4, 5, and 6 levels, and the faulting of the ore-body. It will be seen that all rises and most of the levels are in the lode, and these will sooner or later be affected by any movement of the ore-body, which occurs as the extraction of the ore proceeds, for it must be remembered that the ore-body has been longitudinally sliced along its western wall.

Chinese (Yunnanese) coolies have proved themselves to be the most satisfactory as miners, and many become very efficient. Indian natives are mostly employed for mechanical and other work incidental to mining operations; but, notwithstanding the cheap labour and absence of hoisting and pumping, the cost of mining is approximately 10 rupees (14s. 6d.) per long ton of ore mined.

The labour requires very close and careful supervision, and one of the present difficulties at Bawdwin is the paucity of the supply, which is only sufficient for the present daily needs of ore extraction. No surplus is available for remodelling the workings of the mine or augmenting the tonnage for treatment.

The Yunnanese mostly tramp in from, and back to, Yunnan during the dry season—November to March—some of them walking a distance of 200 to 300 miles each way. Some stay for 12 months or so, while a few have been with the Corporation for a number of years. The majority, however, leave their families in their native villages and only come to Bawdwin for a few months' work, in order to make some money to take home with them. Bawdwin is approximately 3000 ft. above sea-level, but the Yunnanese desire to live at a higher altitude, and on their own plots of land, in their native villages, where their own customs are practised, and where they can grow their necessaries of life. The homes and holdings in their own villages are secure, but the life at the mine is more or less precarious, and, added to this, there is the further complication of a continually changing European staff, which is very prejudicial to the efficient handling and retention of an adequate labour supply.

FIG. 7.


BAWDWIN MINES
PLAN OF 3RD, 4TH, 5TH & 6TH LEVELS
Scale-1inch = 300 feet
▨ Ore Reserves to Decr 31st 1919
▒ Ore Developed since Decr 31st 1919
Nº 3 LEVEL (300 FT)
Nº 4 LEVEL (430 FT)
Nº 5 LEVEL (540 FT)
Nº 6 LEVEL (635 FT)
N Drive
S Drive
S.D.
Rise
Winze
VERTICAL SHAFT
INTERNAL SHAFT Nº 1
Country
North Drive
W X Cut

The Asiatic is capable of giving very loyal service when understood and considered, for he then looks upon his superior with confidence and respect, and will follow him anywhere. The continual changes in the European officers supervising them, unfamiliar with their language, ways, customs, and beliefs, cause the natives to very soon drop to a level of doing just what they have to do and nothing more, until they have saved sufficient money to permit them to leave.

Up to the present the tonnage of ore mined has not exceeded 10,000 tons per month, and, as it is desired to increase this to 30,000 tons in the near future, it will readily be seen that the question of an adequate supply of labour is of considerable importance.

The following figures show the approximate detailed cost of mining. These costs should be reduced when the mine produces a larger tonnage, and are estimated to fall to 8 rupees (about 11s.) per ton.

MINING COSTS

(10,000 tons monthly).

Items.	s.	d.
General expenses		6.5
Survey office		1.5
Repairs and maintenance (including timber)	2	6.0
Lighting		5.0
Compressed air		3.5
Hoisting		3.5
Assaying and sampling		3.0
Supervision	1	0.0
Miners and shovellers	2	0.0
Explosives		6.0
Timber	1	5.0
Steel		4.0
Sundry supplies		2.0
	9	10
Filling stopes	1	9
Tramming	3	6
Total cost of mining	15	1

METALLURGY.

SMELTING WORKS.

In the early days of the Company's career smelting operations were confined to the treatment of the old heaps of slag produced by the Chinese. This slag contained from 40 to 45 % lead, and was railed to Mandalay, a distance of about 200 miles, where a smelting works had been erected on a very convenient site on the banks of the Irawaddy River. This offered many advantages, insomuch as Mandalay is a large town, well supplied with stores, labour, housing, etc., and with an adequate supply of ironstone and limestone near by. Later, these smelters were suddenly moved to Namtu. The real reason is difficult to ascertain, but of those given the most likely is that there was a further discovery, by another company, of old Chinese slag heaps and workings a few miles from Namtu, and it may have been considered that smelting works at Namtu would prove advantageous, especially when all the Bawdwin slag had been smelted, and the works were hungry for fresh supplies. The Bawdwin ore-body had not then been discovered. Whatever may have been the reason, it would appear that these projects yielded nothing but trouble. The new company never prospered, and many disappointments and problems were added to the Burma Mines Ltd.

The site of the present smelting works is on the side of a steep hill, and close to the Namtu River, and, whilst it suited the requirements of the primitive plant during the early days, it offers many difficulties for expansion along modern metallurgical lines. Another drawback is that for six months in the year heavy fogs arise every night from the Namtu River, and, as the smelting works are situated in the Namtu valley, all the gases and fumes emitted from the roasters and smelters, instead of being dissipated into the atmosphere, are condensed in these fogs, which spread and hang over the industrial and residential areas during the night time, and are not dispersed until some hours after sunrise. These moist acid and dust-laden fumes, besides being a discomfort to the inhabitants, cause a rapid erosion of iron and steel work in the vicinity.

The smelting works are served by the Corporation's own railway line of 2-ft. gauge. The top railway yard serves three lines of trestles, two of which command bins for receiving materials for the blast furnaces, whilst the third line is over the bins at the crushing plant, where selected high-grade ore and ironstone flux were crushed. This crushing plant proved to be inadequate for crushing the desired tonnage through ⅜-in. trommel, and was finally used solely for crushing the fluxes. The partially completed concentrating mill, about 3 miles distant, was used for crushing the ore, and better results were obtained after the ore had been reduced to pass through ⅛ screen. The crushed ore, concentrates, and ironstone flux are stored separately in large circular steel tanks, commanding a conveyor belt, which conveys superimposed layers of crushed ore, ironstone, etc., in the proportion of approximately 3 parts of ore and 1 part of ironstone, to the roasting and sintering plants. The blending and mixing is, however, not very efficient, varying over wide ranges, and adversely affecting results.

Originally the roasting was done in 6 circular furnaces of the Godfrey type (revolving hearths and fixed rabble arms), and the partially-roasted mixture was then cooled and sent to the pot-roasting section, which contained 23 large pots, many of which were discarded kettles from the silver refinery, and had been fitted with trunnions and blast-pipes.

Later on the selected ore from the mine became lower in lead content, falling from about 42 % to 37 % of lead, whilst the zinc content increased from 21 to 26 %. It therefore became necessary to treat an increased tonnage in order to maintain the desired output of refined lead and silver. Dwight–Lloyd machines, made in Rangoon, were installed with the utmost despatch to meet these demands and to augment the preliminary roasting capacity. An increased tonnage of ore was treated, the production maintained, and the operating costs were materially reduced, but the metallurgical results were not altogether satisfatory.

The high zinc contents in the mixture sintered, approximating 20 % zinc, caused difficulties in operating, and an appreciable loss of lead. This was rather disappointing, and was probably due to the higher exothermic reaction, resulting from the increased quantity of zinc present. The Australian practice is to have about a quarter of the above amount of zinc in the mixture to be sintered, and no doubt this will be approximated later on, when the concentrating mill is completed and gives the anticipated relief.

The theoretical capacity of the plant is as follows :—

	Units.	Tons per day.
Primary roasting—		
Primary D.–L. machines	3	210
Round roasters	6	240
		450
Final sinter—		
Secondary D.–L. machines	3	220
Sinter pots	23	300
		520
Blast furnaces—		
No. 1		150
No. 2		150
No. 3		80
No. 4		180
		560

The following assay results from the sintered product are interesting, and show that, for preliminary roasting, high lead and silver losses resulted from the double treatment in Dwight–Lloyd machines, and that after preliminary roasting in the Godfrey furnaces, the H.–H. pots, or kettles, gave higher silver and zinc losses than the D.–L. machines, though the lead contents of the sinter remained exactly the same.

The first set of figures are the average results for 1919, when only Godfrey furnaces were used for the preliminary roasting.

Preliminary roasting.	Final sintering.	Ag. oz.	Pb %.	Zn %.
Godfrey furnaces	H.–H. pots	28.9	29.7	18.7
,, ,,	D.–L. machines	29.7	29.7	19.5

The following figures cover the period January to June (inclusive), 1920, and after D.–L. machines were used in the preliminary roasting section :—

Preliminary roasting.	Final sintering.	Ag. oz.	Pb %.	Zn %.
Godfrey furnaces	H.–H. pots	27.5	28.2	18.0
,, ,,	D.–L. machines	28.6	28.2	19.3
D.–L. machines	,, ,,	27.2	27.2	18.9

The sintered ore and ironstone mixture is then sent to the blast-furnace bins per railway cars, and is smelted in the blast furnaces.

The outstanding feature of the blast-furnace work is the high zinc and iron and the low silica and lime content of the slag produced. No difficulty is experienced in running a slag averaging 28.5 % zinc oxide, through the course of a month's continuous operation. On odd days the slag averaged 30 to 33 % ZnO, and ran without giving any trouble. It is, however, most essential that the sulphur content of the sinter does not exceed 3 % if satisfactory furnace conditions are to be maintained, and the formation of crucible crusts and furnace accretions are to be prevented.

The approximate analysis of a satisfactory slag would be :—

Silica, 20 % ; iron oxide, 40 % ; zinc oxide, 30 % ; lime, 5 %.

The limestone required was added to the furnace charge, and the silica was a natural ingredient of the ore.

The slag, when tapped, runs into a small movable forehearth, and overflows into a side-tipping slag bowl, which is mounted on a carriage and hauled to the adjacent dump by a small locomotive on the 2-ft. gauge line.

Unless the sulphur content of the sinter be below 3 %, these forehearths very soon become clogged, and have to be changed.

It was found advisable to rod the crucible from the tap-hole every time the furnace was tapped, in order to prevent the formation of crucible crusts, especially with the high percentage of sulphur on the charge, for it was not possible to keep it reduced to the desired figure of 3 %.

The metallurgical work was not of as high a standard as might have been attained, for it was adversely affected by—

(*a*) Unsuitability of the smelter plant as at present arranged, and especially insufficient crushing and mixing of the charge preparatory to preliminary roasting.

(*b*) Policy of trying to force the construction of a new smelting works on a fresh site, to the detriment of the operating plant.

(*c*) Changing European staff and native labour.

(*d*) Difficulties of operating during the wet season, owing to sickness and shortage of labour, and in obtaining the desired quantities and qualities of supplies.

However, much useful work was done, and when operating and plant conditions are improved, whereby the metallurgist has complete control of all his manipulations, the daily routine of keeping the works running efficiently will be greatly simplified, and better results obtained.

Occasionally, when satisfactory conditions did prevail, the lead in the slag was reduced to below 2 %, but as a rule it averaged about 4 % over a lengthy period.

The sulphur in the sintered material was also too high, and averaged about 5 %.

The completion of the new concentration mill will, no doubt, give considerable relief, for a more uniform product, higher in lead and much lower in zinc, will then be treated, probably with greater ease and much improved recoveries.

Assuming the Corporation is able to overcome its present difficulties, produce and treat 1000 tons per day for six days a week, and does not have any serious set-backs, the following is an estimate of the treatment costs. Particulars of the estimated mill recoveries are given on pages 88 and 89.

LEAD SMELTER AND REFINERY DEPARTMENT.

Basis.—The calculations are made on the estimated tonnage of lead concentrates to be produced per 100 tons of ore concentrated in the new mill.

Blast Furnace.

	CONTENTS.							ASSAY VALUES.				
	Tons.	Ag. oz.	Pb. tons.	Zn. tons.	Fe. tons.	Insol. tons.	CaO. tons.	Pb. %	Zn. %	Fe. %	Insol. %	CaO. %
Lead cons.	37.91	8482	1.5	4.81	.932	.39	—	—	—	6	—	—
Ironstone	6.0	—	—	—3	.60	.72	—	—	—	60	12	—
Limestone	2.0	—	—	—	—	.10	1.0	—	—	—	5	50
Gossan	1.5	—	.07	—	.06	.90	—	5	—	4	60	—
Returns, etc.	6.5	—	—	—	—	—	—	—	—	—	—	—
	55.9	1848	21.57	4.8	5.59	4.11	1.0					
Coke, 15 %	8.4	—	—	—	7.2 FeO	—	—					
Slag ..	19.0	—	3.0	25% ZnO	37.9	21.6	5.2					

Production—

Refined lead (90 % recovery) = 19.41 tons per 100 tons average grade ore milled.

Refined lead (85 % recovery) = 18.33 tons per 100 tons average grade ore milled.

Refined silver (90 % recovery) = 1663 oz. per 100 tons average grade ore milled.

Estimated Cost Sheet (in Rupees).

Basis :—100 tons ore = 18.33 tons refined lead. Allow rupees 14 as being equivalent to £1 sterling.

Mining and Transport—

	Cost per 100 tons Ore. Rs.
Mining 100 tons ore at Rs. 8 per ton ..	800
Transportation, 100 tons ore, at Rs. 2 per ton	200
Milling 100 tons ore, at Rs. 6 per ton	600
Transport 38 tons to smelters, at Rs. ½ per ton	19
	Rs. 1619

Lead Smelting—

Ironstone, 6 tons, at Rs. 9 per ton ..	54
Limestone, 2 tons, at Rs. 5 per ton ..	10
Crushing and sampling, 56 tons, at Rs. 2 per ton	112
Assaying 56 tons, at Rs. ½ per ton ..	28
Roasting and sintering, 56 tons, at Rs. 10 per ton	560
Blast furnace, 56 tons, at Rs. 7 per ton	392
Coke, 8.4 tons, at Rs. 70 per ton ..	588
General labour, 56 tons, at Rs. 3 per ton	168
General expenses, 56 tons, at Rs. 3 per ton	168
	——2080

Mining, transport, and smelting, total	3699 =	Rs. 202 p. t. lead
Refining per ton of lead		Rs. 40 p. t. „
Net cost of lead at Namtu		Rs. 242

Estimated Value of Silver and Lead Produced from each 100 Tons of Average Grade Ore.

(Lead at £25 per ton, silver at Rs. 2 per oz.; £1 = Rs. 14.)

	Rs.	Rs.
Value—		
18.33 tons lead at £25 × Rs. 14	6415.5	
1663 oz. silver at Rs. 2	3326	
		9741.5
Cost—		
18.33 tons lead at Rs. 242 (Namtu cost)	4435	
		4435.0
Value at Namtu		5306.5
Less—		
Realization cost—	Rs.	
18.33 tons lead at Rs. 48 per ton	880	
1663 oz. silver, say	100	
London and general expenses, at 10 % of profit from working account	433	
		1413
Net profit to working account from silver and lead only		Rs. 3893.5= £278 2s.

Equivalent to £2 15s. 7½d. per ton of ore.

Estimated Value of Copper Ore.

(Assuming 100 tons to be treated daily; copper at £60 per ton and silver at 3s. oz.)

The copper ore has the following analysis :—

Silver	17.2 oz. per ton.
Copper	11.0 %
Lead	5.9 %
Insoluble	21.2 %
Iron	17.5 %
Lime	3.0 %
Alumina	6.0 %
Sulphur, about	20.0 %

This would require but a little limestone to flux it, and would produce a lead-copper matte.

A copper smelting plant has, up to the present, not been designed, and when it is it will be necessary to instal a bag house to collect the silver and lead fume from both the blast furnace and converters.

The blister copper is estimated to contain about 160 oz. of silver per ton.

The cost of production will probably equal the estimated value of the copper—*i.e.*, £60—in which case all the silver will be clear profit.

Assuming that 85 % of the silver is recovered in the blister copper, the balance being caught in the lead fume, the net return to the Corporation will be 1462 oz. at 3s. = £219 6s. per day, and, on the basis of 300 operating days a year, it will equal £65,790.

FUEL AND FLUXES.

Coke of fair quality is obtained from India, and costs 70 rupees, equivalent to £5 per ton at the smelters ; but, with all the transhipping and handling it has to receive, there is an appreciable loss in " fines " by the time it reaches the works' bins.

Several finds of good quality coal have been reported in Burma, but on examination have proved, up to the present, to be of very limited extent, in inaccessible places, and in broken country.

Lignite coal of good quality has, however, been found in several districts. The Corporation owns a promising area at Namma, 65 miles from Namtu, and it was originally intended to utilize this coal for fuming off the zinc oxide from the blast furnace slag in reverberatory furnaces, but on closer examination, and with present metal prices, it was decided to leave the question in abeyance for the present.

Limestone of a suitable nature is obtained from surface workings about 4 miles from the smelters, and alongside the main railway line. It costs about 7s. per ton delivered at the smelters.

Ironstone.—The establishing of an adequate supply of ironstone has caused considerable anxiety. Originally it was obtained from a residual deposit overlying limestone at a place called Thondaung, just below Maymyo. The larger pieces were hand-picked from pits sunk on the property. Later on it became necessary to erect a washing plant, but the grade fell, and the villagers near by and below the workings complained of the slime in the water affecting their cultivation areas.

Another ironstone deposit was also worked at Man-Mak-Lang, near Namyao, where there appeared to be a large contact deposit. This was open cut from the surface, and a considerable tonnage was sent to the smelters, but development work from the No. 2 (lower) adit was disappointing, and finally this deposit was abandoned.

Another deposit at Kungkha was exploited, but it also had to be abandoned.

However, when the position was becoming somewhat acute, a residual deposit, which gives promise of becoming extensive, was opened up near Pwekawk, a few miles from Maymyo, and on the Burma Railway Co.'s line.

LEAD AND SILVER REFINERY.

The bars of base bullion from the blast furnaces are loaded on to trucks and lowered down an incline tramway to the head of the refinery.

The improving or drossing furnace is charged with the bullion as required. Originally there was only one furnace of 120 tons capacity, but this is being replaced with two smaller furnaces, the capacity of each being about 80 tons. The dross is skimmed off with iron rakes and sent to the blast furnaces for re-treatment.

The antimony softening furnaces are three in number, and each has a capacity of 35 tons. The lead from the improving furnace is tapped into them, the heat raised, and two skimmings are taken off—the first after 4 hours, and the second when the charge is clean. It takes from 6 to 10 hours to clean a charge, and the skimmings are re-treated in the residue furnace, which has a capacity of about 40 tons. When re-treating these residues about 5000 lb. of skimmings are mixed with 175 lb. of fine coke, and about 20 tons of this mixture are charged into the furnace each day. About 25 % of the lead contents are reduced, recovered, and sent to the improving furnace, whilst the antimonial slag is stacked until sufficient has accumulated for a blast furnace campaign for the production of antimonial lead.

The de-silvering kettles are six in number, and each holds about 30 tons of lead. The lead is tapped from the softening furnaces into the kettles, and subjected to two zinckings. The resulting crusts are skinned, pressed in Howard presses, etc., and passed through a sealed grid to the silver refinery, where they are retorted, cupelled, and the refined silver moulded into ingots of approximately 1000 oz. each, and of a fineness of 998.

The lead from the de-silvering kettles can be syphoned into three refining furnaces, each of which holds 25 tons, and after being refined the lead is run into two moulding kettles, each of which will hold 30 tons. The lead is then drossed and syphoned into moulds, which are then trimmed and the bars are numbered and stacked in the yard preparatory to being railed to Rangoon for shipment abroad.

Silver is a metal much prized by the natives, and they are quite as skilful in acquiring it as they are in working it. Positions in the refinery, therefore, prove very attractive, especially to the Chinese, and it was never difficult to obtain an abundance of labour for its operations. It is, however, a very different matter to detect their thieving, for bribery and corruption are very rife in the East, and, in addition, the Indian is more often than not afraid to interfere with a Chinaman.

The silver refinery was recently enclosed with a closely-guarded brick wall, but this was only partially successful, and, in order to try and stamp out the stealing, it is now proposed to enclose the whole of the refinery, and make each employé change on entering and leaving it.

Bandits or "Dacoits" often make their presence felt in various parts of Burma, and give trouble to the villagers and to the authorities. It is therefore necessary for each consignment of silver—usually of about 75,000 oz.—to be accompanied by an armed European escort of at least five rifles when it is being railed from Namtu for transhipment on to the Burma Railways at Namyao.

The base bullion from the blast furnaces contains about 97 % lead and 110 oz. of silver per ton. The following analyses will show the composition of the products as their treatment progresses:—

Analyses of Refinery Products.

Products.	Ag. ozs.	Pb. %	Zn. %	Cu. %	Sb. %	Fe. %	As. %	Ni. %	S. %
Hard lead from smelters . .	110	97.0	—	—	—	—	—	—	—
Copper dross	61	59.0	6.0	12.0	3.30	.50	2.12	4.20	6.81
Drossed lead	118	98.7	tr.	.15	1.10	.01	—	tr.	tr.
Antimony furnace skimmings	6.0	64.8	.20	.15	18.33	—	1.80	—	tr.
Lead before zincking	112	98.8	.005	—	.05	.008	—	tr.	tr.
Retort bullion	1965	—	1.0	1.9	—	—	—	—	—
Residue furnace slag	1.2	54.7	5.0	.20	23.2	.20	3.50	—	.10
Copper matte + residue furnace	53.6	53.8	tr.	25.6	—	.30	.20	.90	16.0
Nickel speiss	13.6	7.1	—	10.1	7.4	.40	12.60	44.0	.80
Fine silver	99.84%	.0002	—	.0006	—	.0001	(Au	.0004%)	—
Refined lead	tr.	99.9	—	nil	.005	.01	—	(Bi. tr.)	—
Double refined lead	tr.	99.962	tr.	nil	.0006	tr.	nil	nil (Bi .0004)	.0009

CONCENTRATING MILL.

That reliable data might be obtained for the construction of a large and modern concentrating mill, a small experimental one was erected and operated for some time at Namtu, but it is doubtful whether the new mill will fulfil the desired anticipations. The writer is afraid the difficult operating conditions at Namtu have not been fully appreciated. The mill as designed will, therefore, always be difficult to supervise and expensive to maintain and operate.

Its rated capacity is 750 tons of ore per day, and it was originally estimated to cost £270,000, and to be built in two years; but for various reasons both these estimates will be practically doubled.

It is a somewhat difficult matter to give a clear and short description of its flow plan, which is of a very elastic nature. The experimental work was done on ore from the upper levels in the mine. This was oxidized to varying degrees, and the composition of the ore-body varied widely from wall to wall, so that in designing the mill an effort was made to permit of several alternatives in the flow plan, in case they should be needed.

The mill is about 3 miles by rail from the smelters, and consists of a detached rock-breaker station, 3 coarse crushing and jigging sections, 2 tabling sections, and a flotation plant. This latter is situated across the main railway line, and is housed in a separate building. All the foundations of the mill are of concrete, and the superstructure of timber covered with corrugated iron.

At the head of the mill there are coarse ore bins of about 1500 tons capacity. These are commanded by the Corporation's railway line, by which the ore is brought from the mine ore bins at Tiger Camp. The ore bins discharge on to inclined conveyor belts, which in turn command the breaker station, situated on the same level.

The breaker station consists of one large and two smaller gyratory breakers (No. 5). These deliver the ore, crushed to about 1½ in., on to another inclined conveyor belt, which crosses the incoming ones on the reverse angle, and conveys the ore back to the fine ore bins, adjoining and in front of the coarse ore bins.

Under the fine ore bins are a number of conveyor belts. These feed three main belts running at right angles to them, which, after passing over automatic weighing machines, convey the coarsely-broken ore to the three coarse-crushing sections of the mill proper.

Each section contains two elevators, each 24-in. by 50-ft. centres. One of these receives the coarsely-broken ore and elevates it to a shaking screen, trommel, or Mitchell screen, according to what is in operation at that particular section. They are fitted with ⅛-in. screens, and the rejects pass back through the rolls, 36 in. x 18 in., and of the Broken Hill type, to the bottom of the elevator again. The fines from the screens are fed on to Callow screens, covered with 22-mesh wire screens. The minus 22 passes to the Wilfley roughing tables, and the plus 22 feeds a double compartment Hartz jig, fitted with ¼-in. screens on two hutches, and 3/16 in. on the third. The tailings from the jig can be crushed in one of four tube mills, each 10 ft. by 5 ft., the discharge from which is elevated to Callow screens of 22 mesh. The undersize passes to Allen cones, of which there are sixteen, and thence to the primary Wilfley tables.

The oversize from the Callow screens, together with the discards from the Wilfleys, passes to one of 4 more 10-ft. x 5-ft. tube mills, where they are crushed to pass a 36-mesh Callow screen, thence via more Allen cones to 24 Wilfley tables, whilst the slimes pass on to the flotation plant.

The primary tables make a lead concentrate, middlings, and tailings. The secondary set makes four products—lead concentrates, zinc concentrates, a zinc-lead product, and tailings.

There are also 40 Deister tables, commanded by Dorr classifiers, for re-treating the finer and intermediate products.

Both the Wilfley and Deister tables are arranged on two floors, one above the other.

There are 96 tables in all, 56 being Wilfleys, 16 of which are used for roughing, 16 for primary, and 24 for the secondary work.

The elevators, of which there are 32, are almost vertical, and of about 50-ft. centres, and each has a large concrete pit for a "boot." The buckets vary in size from 18 in. to 24 in., and the elevators are arranged in five rows right across the main mill building. This makes it impossible to see for any distance over the mill from any particular point. Another disadvantage is that all launders are underneath the floor, and, together with leaks and losses, they command a big tunnel system underground, which was to have been one of the features of the mill. Unfortunately, its grade is insufficient to carry off losses and spillings, and this tunnel system has caused an undue congestion and complication of the launders. As many as four launders are superimposed in places, each one carrying a different grade product.

The lead product from the jigs and roughing tables runs into shovel wheels, and thence per an 18-in. conveyor belt to the storage platform near the railway line. The fine concentrates from the other tables are run on to a draining belt, which delivers into three wooden draining vats or paddocks, and are loaded into railway trucks.

The mill produces two grades of slimes—

(1) High-grade from the shovel wheel, which is run through Dorr thickeners on to a 12-ft. x 8-ft. Oliver filter, commanding a belt conveyor which feeds a wood-fired drier. This dried slime is then sent to the smelters.

(2) The low-grade slime is sent to the flotation plant, and, after being thickened in a Dorr thickener, is mixed with a weak solution of soda ash, a very little eucalyptus oil, and floated in four horizontal flotation boxes. These can be run in either series or parallel, or the product of the first three boxes can be refloated in the fourth box.

It is further proposed to instal a cascade plant for floating the middlings.

It was also proposed to dewater all concentrates on 12-ft. x 8-ft. Oliver filters, but, as the high-grade slime still contained 17 % of moisture after leaving the filters, a trial vat with a filter bottom and vacuum pump attached was erected, and promises to supersede the Oliver filters for collecting and drying the lead products.

All the lead products are sent to the smelters for immediate treatment. The zinc concentrates are stored for shipment when that becomes profitable.

The Corporation intended to erect a zinc distillation plant of 25,000 tons capacity near Tata's iron and steel works at Jamshedpur, in India, for the treatment of a portion of its output, but, after partially erecting the plant, construction work was suspended.

The machinery for the mill is all belt-driven by independent motors.

The following figures show an estimated mill balance-sheet, compiled in September, 1920, for the treatment of average grade ore as shown by the mine reserves. Actual results from the mill indicate that the anticipated recovery of 80 % lead will be exceeded, and the residue will be reduced to under 5 % lead content.

ESTIMATED MILL BALANCE SHEET, 1/9/20.

Calculated on Average Grade of Ore in Mine.

Ref. No.	Weight. Tons.	Assay.					Metal Contents.					Percentage Contents.				
		Ag. oz.	Pb. %	Zn. %	Fe. %	Insol. %	Ag. oz.	Pb. tons.	Zn. tons.	Fe. tons.	Insol. tons.	Ag.	Pb.	Zn.	Fe.	Insol.
1. Mill feed ..	100	23.5	26.5	18.7	5.0	31.6	2350	26.5	18.7	5.0	31.6	100	100	100	100	100
2. Lead cons...	27	48.8	57.5	13.0	4.5	7.0	1320	15.5	3.5	1.2	1.9	56	58.5	20.3	24	6.0
3. Middlings ..	10	28.0	33.0	30.0	7.5	3.4	280	3.3	3.0	.75	.34	12	12.4	14.4	15	1.1
4. Zinc cons. ..	15	12.7	10.0	46.0	7.0	3.5	190	1.5	6.9	1.0	.8	8	5.7	36.9	20	2.5
5. Slime	15	25.3	28.0	18.0	5.0	31.6	380	4.2	2.7	.75	4.7	16	15.8	14.4	15	15.0
6. Tailings ..	33	5.5	6.0	7.8	3.9	71.6	180	2.0	2.6	1.3	23.86	8	7.6	14.0	26	75.4
							23.5	26.5	18.7	5.0	31.6	100	100	100	100	100

RE-TREATMENT OF MIDDLINGS.

Ref. No.	Weight. Tons.	Assay.					Metal Contents.					Percentage of Original Ore Content.				
		Ag. oz.	Pb. %	Zn. %	Fe. %	Insol. %	Ag. oz.	Pb. tons.	Zn. tons.	Fe. tons.	Insol. tons.	Ag.	Pb.	Zn.	Fe.	Insol.
3. Middlings (feed)	10	28.0	33.0	30.0	7.5	3.4	280	3.3	3.0	.75	.34	12.0	12.4	14.4	15.0	1.1
2A. Lead cons. ..	4.8	46.6	55.0	12.0	7.7	1.9	224	2.64	.6	.37	.09	9.5	10.0	1.4	7.4	.3
4A. Zinc cons. ..	5.2	10.8	12.7	46.0	7.3	4.8	56	.66	2.4	.38	.25	2.5	2.4	13.0	7.6	.8

RE-TREATMENT OF SLIMES.

Ref. No.	Weight. Tons.	Assay.					Metal Contents.					Percentage Contents.				
		Ag. oz.	Pb. %	Zn. %	Fe. %	Insol. %	Ag. oz.	Pb. tons.	Zn. tons.	Fe. tons.	Insol. tons.	Ag.	Pb.	Zn.	Fe.	Insol.
5. Slime feed ..	15.0	25.3	28.0	18.0	5.0	31.6	380	4.2	2.7	.75	4.7	16.0	15.8	14.4	15.0	15.0
2B. Lead cons.	6.1	49.8	55.0	12.0	6.0	7.0	304	3.36	.7	36	.4	13.0	12.6	3.7	7.2	1.2
7. Residue ..	8.9	8.5	9.4	22.0	4.4	49.1	76	.84	2.0	.39	4.3	3.0	3.2	10.7	7.8	13.8

COMBINED RECOVERIES.

Ref. No.		Metal Contents.						% Recovery.				
		Tons.	Ag. oz.	Pb. tons.	Zn. tons.	Fe. tons.	Insol. tons.	Ag.	Pb.	Zn.	Fe.	Insol.
2	Lead concentrates	27.0	1320	15.50	3.5	1.20	1.90	56.0	58.5	20.3	24.0	6.00
2A	,, ,, ...	4.8	224	2.64	.6	.37	.09	9.5	10.0	1.4	7.4	.36
2B	,, ,, ...	6.1	304	3.36	.7	.36	.43	13.0	12.6	3.7	7.2	1.20
	Total lead concentrates ..	37.9	1848	21.50	4.8	1.93	2.39	78.5	81.1	25.5	38.6	7.56
4	Zinc concentrates	15.0	190	1.50	6.9	1.00	.80	8.0	5.7	36.9	20.0	22.5
4A	,, ,, ...	5.2	56	.66	2.4	.38	.25	2.5	2.4	13.0	7.6	.8
	Total zinc concentrates ..	20.2	246	2.16	9.3	1.38	1.05	10.5	8.1	49.9	27.6	23.3
6	Tailings	33.0	180.0	2.0	2.6	1.3	23.86	8.0	7.6	14.0	26.0	75.4
7	Residue and slimes	8.9	76.0	.84	2.0	.39	4.3	3.0	3.2	10.7	7.8	13.8
1	Original ore	100.0	23.5	26.5	18.7	5.0	31.6	100.0	100.0	100.0	100.0	100.0

RAILWAYS.

The Burma Railway Company, a more or less Government concern, uses a line of metre gauge, from Rangoon to Myohaung Junction, near Mandalay, the distance is 383 miles. It is practically level, and the journey occupies about 20 hours.

The next stage, to Maymyo, is only 37 miles, but it takes 5 hours to negotiate, as there is a very sudden rise from the plains to the mountains. Maymyo being about 3600 ft. above sea-level, the railway has to zigzag, on a grade as much as 1 in 25 in many places, up the face of a high cliff. Locomotives of the "Mallet" type are used. They weigh 95 tons, and will haul 120 tons up this grade ; but, as the trains are heavier than this, they are usually assisted with one or two "pushers." Except for these "ghats," the ruling grade is 1 in 40, and on this the "Mallet" engines will haul 250 tons. As it is dangerous to run the trains after dark, passengers have to wait in Maymyo over night and proceed next day to Namyao, the 124 miles taking 9 hours to negotiate. At one point the train has to pass over the Gokteik viaduct. This is built of steel, is 1620 ft. long, and weighs 4200 tons. The train takes about two minutes to pass over it, and it is 320 ft. above the foundations, which are built on a natural bridge of limestone, 850 ft. above the bed of the river.

Namyao, 548 miles from Rangoon, is the junction of the Burma Railway Co.'s line and the Corporation's single 2-ft. gauge line. Namtu is 35 miles from Namyao, and in making this journey the train passes over two mountainous ranges, and negotiates steep grades (3 %) and several hairpin-bends of 60 ft. radius.

The Corporation's main line locomotives weigh about 35 tons, are wood-fired and able to haul 50 tons each way. Generally they take about 12 hours to make the round trip—Namtu to Namyao and back.

Each waggon is fitted with a hand brake, and a native brakesman is deputed to look after the brakes of each two waggons. This is a source of weakness, for if the brakesmen are careless a derailment can easily occur on the steep grades and sharp curves, and, apart from the material damage to the line, rolling stock, and freight, it causes a temporary blockage of the line and reduces its carrying capacity. The engines are driven by either Europeans or Anglo-Indians (Eurasians).

Usually about five trains run to and from Namyao each day. Bawdwin is 12 miles by rail north of Namtu; but, as all the ore is railed from Tiger Camp, the station below Bawdwin, only one train a day reaches that centre. The line from Namtu to Bawdwin offered many engineering difficulties. Its route was along a narrow valley with precipitous hills on either side and a fast-running river in the bottom, over which the railway line has to continually cross. At one point it is necessary, in order to obtain the desired elevation, to make the line spiral (corkscrew), and at another to have two zigzags. During the wet season the line is at times washed away in places, and all traffic interrupted until repairs can be effected.

Transportation in and around Namtu is quite a problem, for Namtu is in a narrow valley, the sides of which are steep, and several hundreds of feet in height. The native bullock carts only carry small loads, and have an aversion to working in the heat of the day. As the climate is "seasonal," a heavy strain is thus thrown upon the railway to cope with the extra demands for inter-department supplies during the vigorous activities of the dry season.

The following figures enable one to form a better idea of the demands made on the railway. The tonnage of all material handled on this narrow-gauge single line during 1920 averaged 35,682 tons per month, whilst the ton miles hauled averaged 491,573 monthly. The average cost was equivalent to but 3d. per ton mile, though included in this cost are items for large improvements in the maintenance of both rolling stock and permanent way (rail renewals, etc.)

The Burma Railway Co. held out very strong hopes that they would extend their metre gauge line from Thapali to Namtu, which would have made a very great saving in freight for the Corporation. The necessary survey and estimates were made, but later, on the grounds of financial stringency, the railway company advised they were unable to proceed with the project, which, exclusive of the extra rolling stock required, was estimated to cost £700,000 (about £20,000 per mile). This, naturally, was a great disappointment to the Corporation, for its present railway, besides necessitating expensive double handling, is overtaxed, and will require additional engines and rolling stock to cope with its present requirements. However, with a slight regrading of the main line, and additional rolling stock, and under its present capable manager, the railway will be able to cope with all the traffic of the immediate future.

The railway department has a well-equipped running shed and carriage and waggon shop, where almost any class of repair can be expeditiously effected, and where many covered waggons, open trucks, and a saloon carriage have been built.

Power.

There are three main power-generating plants—one just below the slag dump at the smelters, where there are installed Babcock and Willcox boilers, and two turbo-blowers for serving the blast furnaces. The boilers are oil-fired, for the railway was unable to cope with the extra quantity of wood required by these boilers.

On the opposite side of the river, and near the workshops, there are two large Diesel engines, direct-coupled to two generators, each capable of generating 850 kw. These engines have given very satisfactory service. The kw. hours per month have averaged about 500,000, and cost a halfpenny per kw. hour. This installation may be considered as the central power plant, for it contains the main switchboard and necessary transformers for receiving and stepping down the 33,000 volts received from the Mansam hydro-electric plant. The electric power is also distributed to Bawdwin for working the air-compressor, workshops, hoists, etc., the concentrating mill, smelter plant, refinery, the machine shop, etc., and for lighting Namtu.

The third plant has just been completed at Mansam, where power is generated hydro-electrically. The installation includes 3 turbines supplied by Boving's. Each has a horizontal shaft with a single runner, and with the water inlet rising from the supply pipe. It passes directly under the turbines, which should develop 3000 h.p. at 750 revolutions, but with a maximum efficiency of 84 % when developing 2500 h.p. Direct-coupled to the turbines are General Electric Co.'s generators, 50-cycle, 3300 volts. The current generated is stepped up to 33,000 volts and transmitted to Namtu, a distance of about 20 miles in a direct line. This route lies across some high, mountainous country, and in the wet season, when there are severe electrical disturbances, the transmission line is sometimes struck, and the switch thrown out, thereby causing inconvenience at the points of consumption owing to the cessation of supply. The accompanying illustrations (Plate IV.) show the Mansam Falls, with the generating station, and the steel pipe running down the hill to it, and from the teak-wood fluming which brings the water from the intake, about three-quarters of a mile away. The fluming is only capable of carrying enough water for two generators, and during the dry season the flow falls so low as to only permit one set being operated.

WORKSHOPS, ETC.

The workshops, together with the foundry, cover a large area, are very complete, and are capable of effecting all necessary repairs to the existing machinery, and also of making a large quantity of the new cast-iron and steel-plate work for new constructional purposes.

ACCOUNTS, TIMEKEEPING, ETC.

The Corporation's activities are spread over a widely-scattered area, which renders it difficult to properly control the clerical records, timekeeping, etc., especially as most of the work is done by native contractors, who employ their own labour, and are about as tricky as the proverbial waggon-load of monkeys.

The other employés—" regular hands "—are paid by the Corporation's paymaster. His position and that of the timekeeper carry with them a great many difficulties, and also many temptations, for, as previously stated, bribery and corruption are rife in the East. The natives are of mixed races and languages, and are extremely well versed in all the tricks imaginable for furthering their interests to the detriment of the Corporation.

Each native, when engaged, is registered, and receives a numbered disc of metal and also a specially-printed time card, which is renewed each month. He has to deposit his disc at the time office as he enters the gate going to work, and later gives his time card to the foreman, who initials it in the space opposite that particular day. The timekeeper's clerk makes his entry in his book from the discs, and then checks them with the time cards later on in the day, and as the employé leaves his work his time card is handed back to him. In addition to the " regular hands," there are a number of " casuals," who are " stop-gaps," employed by the day, and they often run into very large numbers. They are booked on through the time office, and when they leave work they are given a ticket representing the amount each has earned, and which can be cashed each day at the head office. A timekeeper is also deputed to visit them each shift and see that they are actually at work.

The position is further complicated by the fact that the Corporation has a large grocery store department, and supplies a large variety of goods, especially foodstuffs, on a kind of credit system. This is done by the employé making application at the time office and presenting his time card. Authority to purchase up to a certain amount, and within his actual earnings, is then given in writing, and when the purchase has been made the time office is advised, and the amount deducted from the employés pay. This would be much simpler if the natives were all of one race and language ; but with the admixture of races and languages, and the numbers that have to be served, which run into several thousands, it becomes a very difficult matter to accurately record the whole of the transactions.

The system in the accounts, costing, and paymaster's offices are all based on sound lines, though perhaps somewhat unnecessarily complicated. A great deal of extra booking is also necessary for each officer's account, owing to the distance from a bank, and the use of the "Chit" credit system.

The leading positions in this department are held by Europeans. The better-class Chinese, Burmans, and Indians from Madras, however, make very good clerks and assistant acountants. A few have an excellent capacity for work, and are quite equal, if not superior, to the European.

HEALTH AND SANITATION.

The hospital and sanitary department is quite an important branch of the Corporation's sphere of operations, as it is responsible for the health of the employés, European and Asiatic, and in order to protect them it must carefully guard the water supply and also watch the health of the community surrounding and arriving in Namtu. This is a very difficult matter, for the natives have not even the faintest idea of sanitation, and as soon as a bad epidemic breaks out in any native village many of the inhabitants generally leave and go elsewhere, carrying the disease with them.

There is always a constant stream of both food supplies and natives coming into Namtu from other centres. The Sanitary Department has therefore to be constantly on the look-out to see that no epidemic creeps in with them. Suspected cases are isolated, there being several isolation camps always in readiness.

The last bad epidemic was an outbreak of cholera in 1918. It caused the Corporation a very anxious time, and cost a large sum before it was stamped out.

One of the European doctors recently contracted typhus fever through attending some newly-arrived coolies who had cleared away from their native village where the inhabitants were dying in large numbers. Typhus fever is contracted through the lice from an infected person biting one, and its mortality is about 40 %. The doctor fortunately recovered.

The actual loss of time and earning power amongst the natives through illness is also an important item, for it runs into a very large sum, and causes considerable inconvenience to the operating staff.

Formerly Asiatic diseases were very prevalent, especially malarial fever, but with the gradual clearing of the jungle, and better sanitation generally, the living conditions at Namtu have much improved, though at the outlying districts, where subsidiary operations are carried on, there is room for much improvement.

Namtu is, roughly, 2000 ft. above sea-level, and the wet season is somewhat trying, especially to anyone with a constitutional weakness. To keep fit, one should take the ordinary precautions, plenty of exercise, and be very moderate in both eating and drinking.

STAFF AND LABOUR.

Labour difficulties are undoubtedly the chief problems of the present-day management, and the reasons, so far as one can judge, are rather interesting to follow.

The company was weak financially when it started smelting at Namtu, and the expense in equipping and establishing itself were very heavy. Lead was at a low price, and funds soon became exhausted. However, hoping against hope, the company continued operations, but was unable to pay its staff and coolie labour in full for months at a time, and on occasions the feeling became very high. Finally, pressure was brought to bear on the company, and with a view to re-establishing the prestige of the white man in the eyes of the native, a very important matter in the East, sufficient money was obtained from London to liquidate all liabilities, pay the staff and coolies in full, and permit a fresh start being made. Since then the company has had to earn the large sums of money it required to develop the mine and to further establish and expand its operations. In so doing, and as the mine developed, its control passed through the hands of various groups of financiers with different ideas of policy. This gave rise to conflicting interests amongst those directing the affairs of the company, and, unfortunately, this was reflected throughout the whole of the organization. It has been largely responsible for the fact that, during the past few years, there have been at least eight resident managers, only one of whom, it is stated, voluntarily resigned through ill-health. The advent of each new manager causes an inevitable staff upheaval and some changes in its personnel. Most of the managers have come from the United States, and formerly the majority of the staff were Americans; but lately their numbers have decreased, and Britishers now predominate. This also further complicated matters, and created feelings of antagonism amongst a number of the staff officials.

These unsettled conditions naturally demoralized the staff, and this, in turn, was reflected upon the native labourer.

The Shans and most of the Burmans have no incentive to accumulate and therefore will not work regularly; consequently, nearly all the labour has to be obtained from either India or China, directly or indirectly.

Change of habitation and occupation is, however, a very serious matter to the Asiatic. The Corporation must appreciate the difficulties and view-point of its labourers, and assist them to work and settle in and around Namtu. This is especially important as regards the miners for Bawdwin. The natives, however, frequently find themselves handled by unsympathetic "new chums," and the consequences have been that when they realized the conditions were unsatisfactory they moved elsewhere as soon as it suited them to do so, and naturally they did not carry away a good impression nor speak favourably of Namtu to their fellow-countrymen. It therefore gradually becomes more and more difficult to obtain an adequate supply of labour.

This is most undesirable from an operating point of view, and also from a question of policy. The seed of industrial unrest has been sown amongst the Asiatics, and especially with the Indian. In Burma, too, the Young Burman Association is working for the advancement of the Burman's scope of activities. It therefore behoves the industrial organizations in the East, and especially the Burma Corporation, in its isolated geographical position, to be in close touch with the native labour, and endeavour to lead and guide it, and so reduce as much as possible any causes for dissension likely to lead to labour troubles.

The number of natives employed varies ; in the wet season it is less than in the dry season, and an approximate average of those regularly employed will be as follows :—

Department.	No. employed.
Smelters	900
Refinery	200
Concentrating mill	300
Bawdwin mines	1000
Various sub-departments (workshops, power, etc.)	500
Railway	430
General work (housing, sanitary, etc.)	570
Total	3900

These figures do not include the labour for outlying properties, nor for forest work and timber and firewood cutting in the dry season. They would probably total another 2500 hands.

The lowest wage is 1s. 2d. per day of eight hours for unskilled labour, and rises to about 5s. for skilled labour.

The natives are, in many instances, capable of performing delicate and intricate routine work, providing it is not arduous, but they must be closely and constantly supervised, and, in addition, the Corporation has to house them and look after their sanitary arrangements, water supply, etc., free of charge.

As the tonnage mined and treated would only average about 10,000 tons per month, it will be seen that native labour is really not a cheap labour to employ.

It has not been found advisable, except in one or two very special cases, to place natives in a position of authority, for, besides becoming too presumptuous, it is exceptional to find one who can rise superior to the temptation which surrounds him for making money and neglecting his employers' interests. This applies more to the Indian than to the Chinaman.

Some foremen have been reported to have made the coolie pay to join their working gang, an additional sum every month to hold his job, and finally a further amount for obtaining a reference when he leaves.

As has been previously mentioned, it is desired to increase, in the near future, the tonnage of ore mined to 30,000 tons per month. It will therefore be seen why so much stress has been laid on this question of studying the natives and gaining their confidence in order to secure an adequate supply.

American interests are reported to have secured valuable silver-lead properties in China, and just over the border of Nothern Burma. Should active operations be established there, it is very probable that further difficulties will arise, and the supply of labour for Bawdwin and Namtu will be still further affected.

The Corporation's registered office has recently been transferred to Rangoon from London, and a local board of directors was appointed. Unfortunately, this has not proved a success, and has not helped to overcome the existing difficulties.

A point worthy of consideration is that an officer, to be of full value to the Corporation, must have a knowledge of the languages and customs of the countries of the natives he is working. If, however, he continues in the service of the Corporation in an isolated place like Namtu, each year of residence cuts him off from his acquaintance and contact with his home group of operators. Many realize this, and that conditions are unsatisfactory, and tender their resignations as soon as they are in a position to do so. These are termed " calendar tickers," and to a number of them the temptation to augment their salaries by subterranean methods is very great.

An officer, to continue in the service of the Corporation, must be made to feel that his agreement is fair and equitable, and that his appointment is to be sufficiently permanent and well enough remunerated to permit him to save an adequate amount to pay for the lengthy and necessarily expensive holidays, and also allow him to provide for his old age.

Namtu is an isolated station, and every Britisher has to be a member of the Indian defence force, and do a certain number of drills in case of emergency.

The living conditions at Namtu will always be difficult and trying to European officers and their families, even when climatized residents, and it takes the best part of two years before a new arrival becomes really familiar with local conditions and of full use to the Corporation. It therefore follows that frequent changes in the management, which have been the rule up to the present time, give neither the Corporation nor staff a really fair chance.

Improved facilities for amusement, diversions, and holidays are also needed to keep the inhabitants of Namtu in a healthy frame of mind. There is a very good club house there with a concrete tennis court attached. Golf links were laid out on the racecourse, where sports and races were held at Christmas time; but the course was subsequently built over, spoilt, and not replaced.

Summary.

The mine and treatment works are situated in the Shan States, near the Chinese border. The mine was first worked by the Chinese as a commercial exploitation, and without political significance, from 1412–1868 A.D. Large quantities of ore were mined and treated by them for the recovery of its silver in very primitive furnaces, but the excellence of their methods, all things considered, is generally conceded.

The first European company was formed in 1903 to treat the 150,000 tons of lead slag left by the Chinese. It was not until 1913 that the faulting of the ore-body and the existence of the large bodies of rich silver–lead–zinc ore were fully understood and realized. Development work has been actively carried out since then, but the estimated tonnage of the ore reserves does not include any allowance for the necessary loss in extracting the ore.

Development work below No. 6, at present the bottom level, requires early attention, for the lateral extensions of the "Shan" and "Chinaman" ore-bodies have been defined, and No. 6 level is only 55 % of the area of No. 4 level. The nature of the ore-bodies below No. 6 level are therefore full of interest as affecting the life of the mine. The best grade ore has been selectively mined. This necessitated an undesirable system of mining, and sliced the body along its western wall. Careful attention should be paid to the safety of the mine workings if a steady output is to be maintained. Fresh levels will need to be driven in the country, and parallel to the lode, and new rises made to the surface, and provision for picking up the sill floor timbers, if the maximum amount of ore is to be extracted.

The metallurgical treatment of the ore does not offer any real difficulties. It will concentrate fairly easily, and give satisfactory recoveries, and slags from the blast furnace have contained up to 30 % zinc oxide without causing any running troubles.

A new and modern smelting plant, to include the recovery of zinc from the blast-furnace slags by fuming in reverberatories, was sanctioned, and construction started; but on further consideration its completion was held in abeyance.

The working of the existing plant, as at present arranged, may cause the operating staff an undue amount of anxiety until the weaknesses can be remedied.

Labour problems, both European and Asiatic, are undoubtedly the chief difficulties of the present-day management, and they will need to be overcome before there can be any material improvement in the output, which it is desired to increase from 10,000 to 30,000 tons per month.

Namtu is almost 600 miles inland from Rangoon, in an isolated position, and but 40 miles from the Chinese border. Local conditions are somewhat difficult, and need to be carefully studied and understood. This takes time, and sufficient attractions should be offered to induce the staff to remain for a lengthy period of service.

The Corporation, besides being a mining, metallurgical, and railway concern, is also a more or less unchartered municipality, for practically every building shown in the illustrations (Plate V.) belongs to the Corporation, and is administered by it. This throws heavier burdens on the resident manager, who, in addition to his responsiblities incidental to winning and treating the ore, has to supervise the Corporation's grocery store, bakery, butchery, water supply, roadmaking, bridges and footpaths, sanitation, lighting, and the housing of all employés, native and European.

It is a somewhat difficult and delicate matter to fully describe and summarize the operations of the Corporation, and a great deal of detailed particulars have been omitted, but the information given has been compiled with the object of interesting Australian mining engineers, as distinct from outside shareholders and speculators.

It is not the intention in this article to forecast the probable financial results of mining and smelting operations, which, up to date, have not been productive of dividends, all the profits having been absorbed in developing and establishing the workings of the properties.

In conclusion, however, it may be mentioned that the Burma Mines Ltd. recently increased its capital to £14,000,000 by issuing 14 shares in the Indian Company (re-named the Burma Corporation Ltd.) for each of the 1,000,000 shares in the Burma Mines Ltd. It was also considered advisable, in order to liquidate liabilities and provide ready money, to raise £1,000,000 by debentures at 8 % interest, from which it will be seen the anticipated activities of the Corporation are of considerable magnitude.

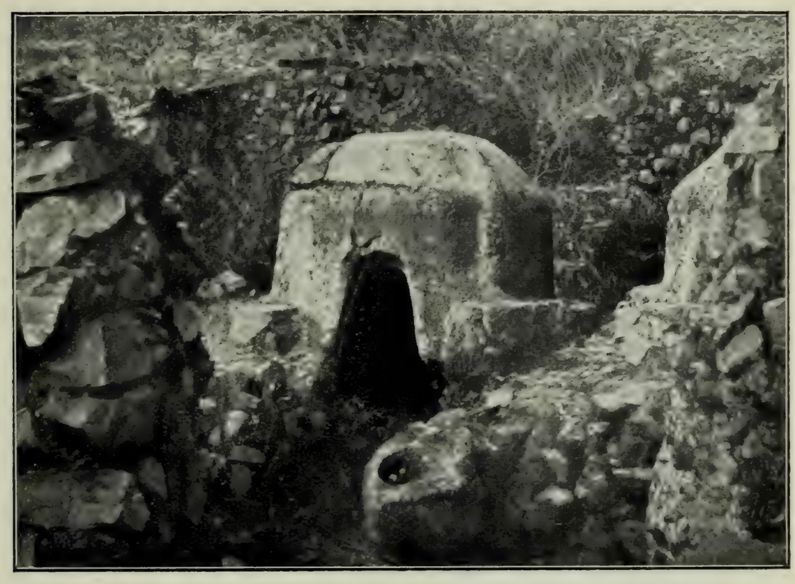


Fig. 1.
Chinese Cupel Furnace, Bawdwin.

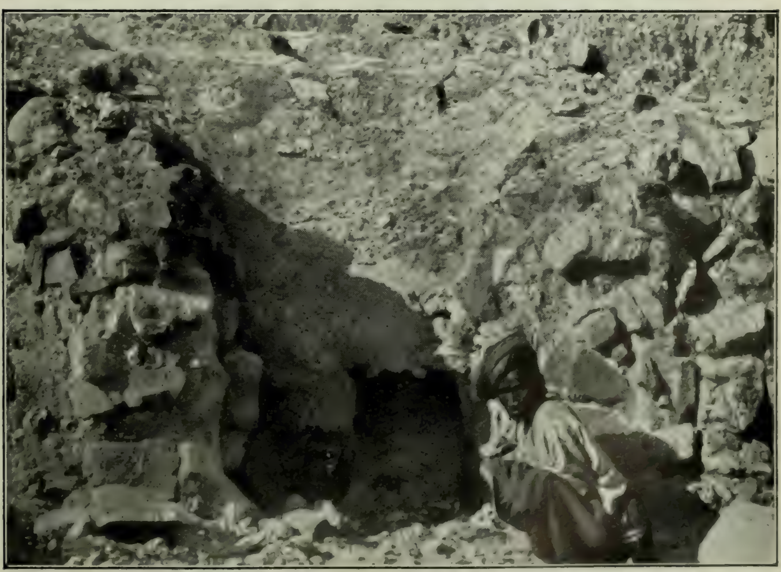


Fig. 2.
Chinese Smelting Furnace, Bawdwin.
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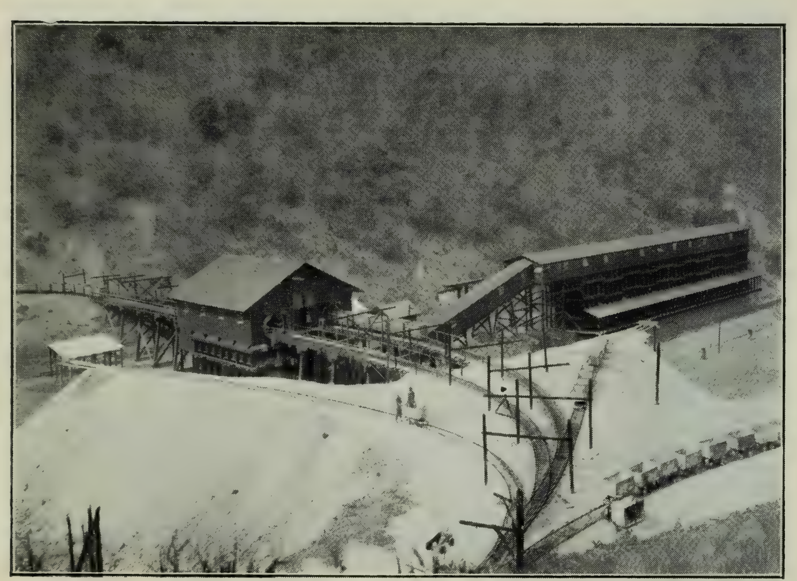


FIG. 1.
Tipple Plant commanding Mine Storage Ore Bins.

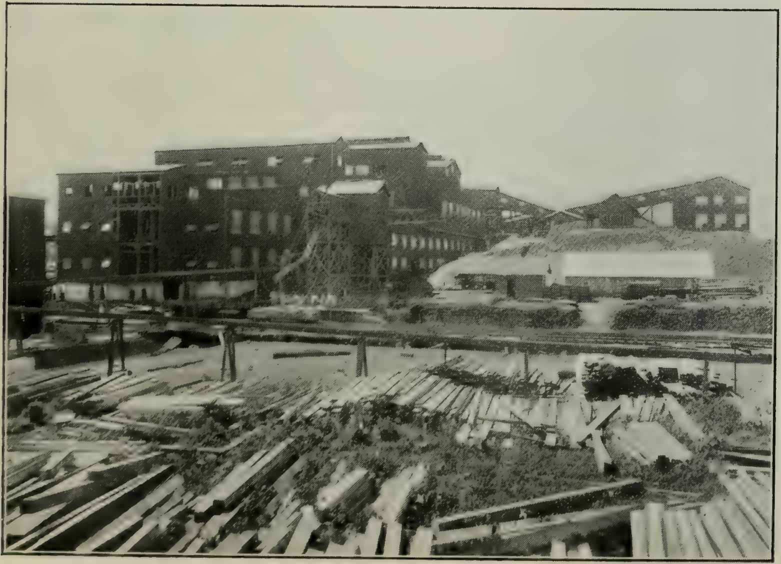
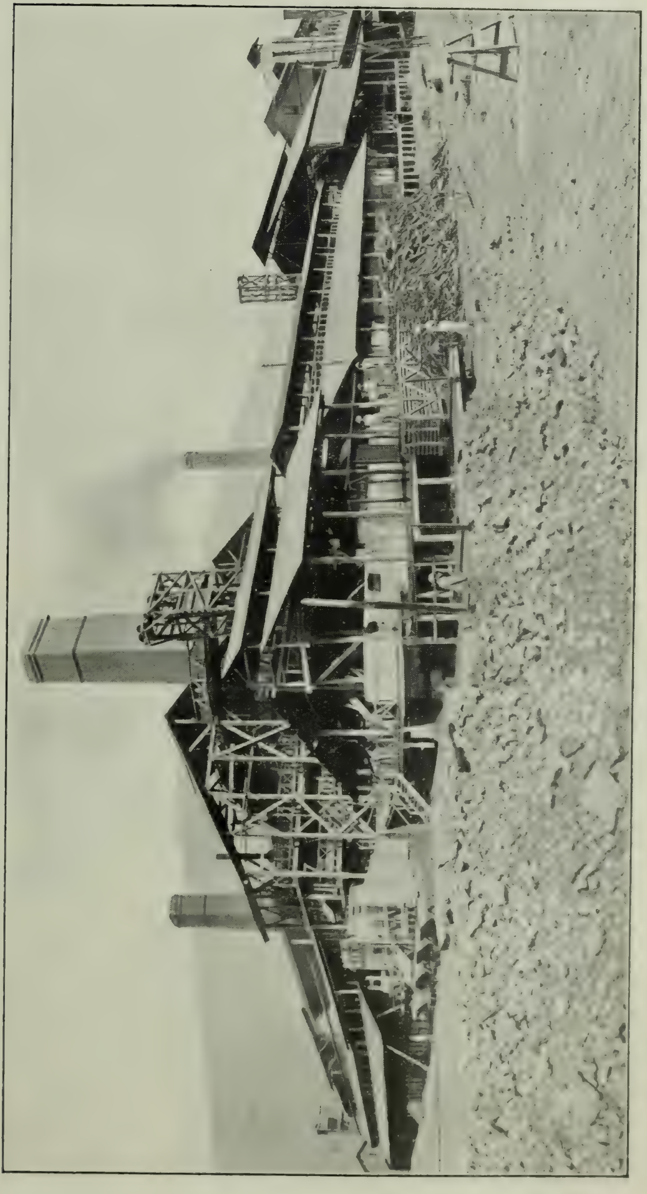


FIG. 2.
Concentrating Mill.



General view of Namtu Smelter.

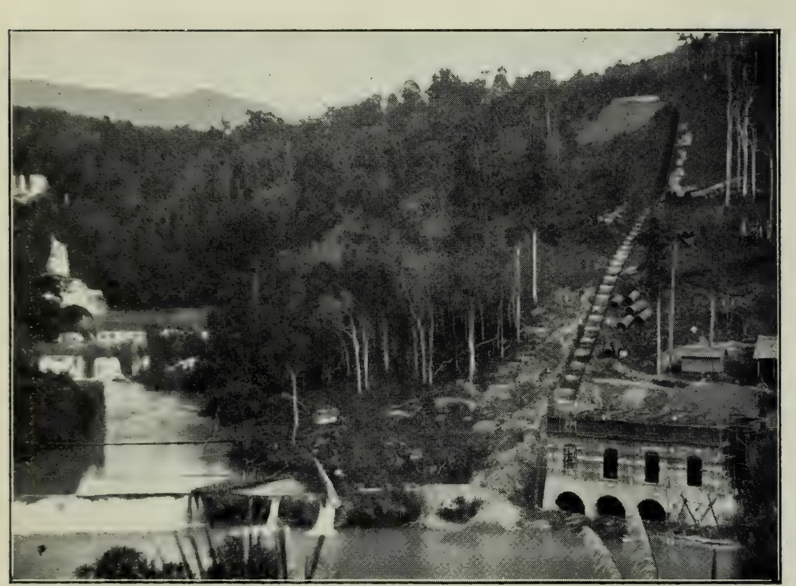


FIG. 1.

Mansam Falls.

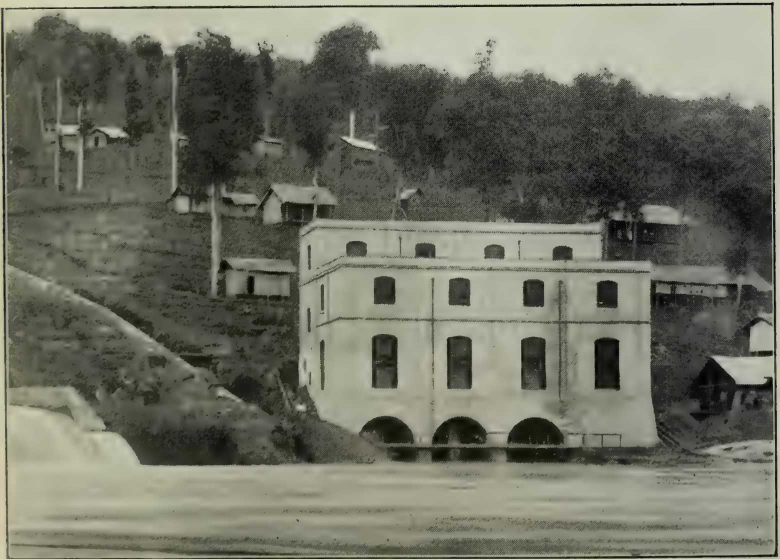


FIG. 2.

Mansam Hydro-Electric Plant.

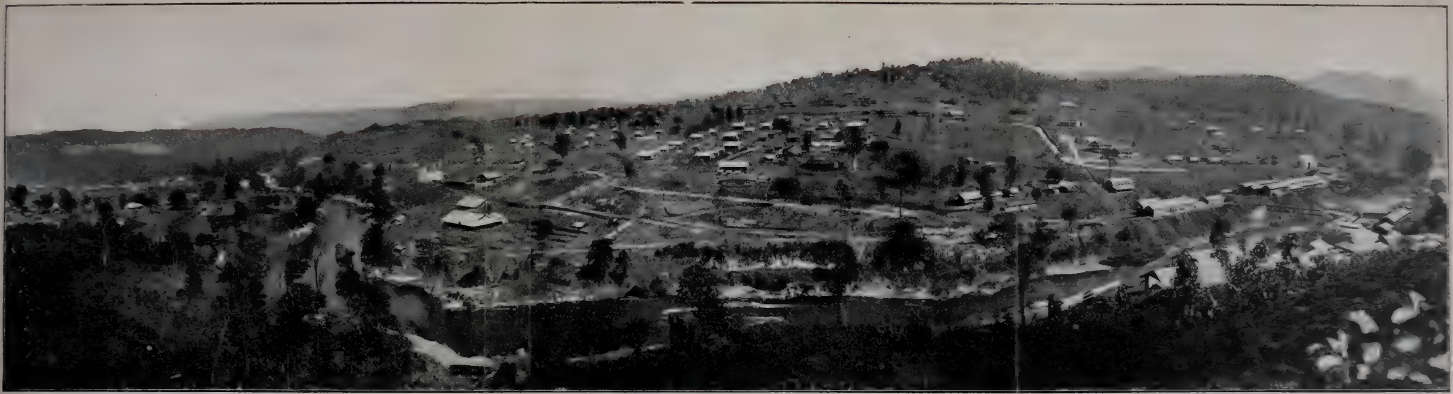
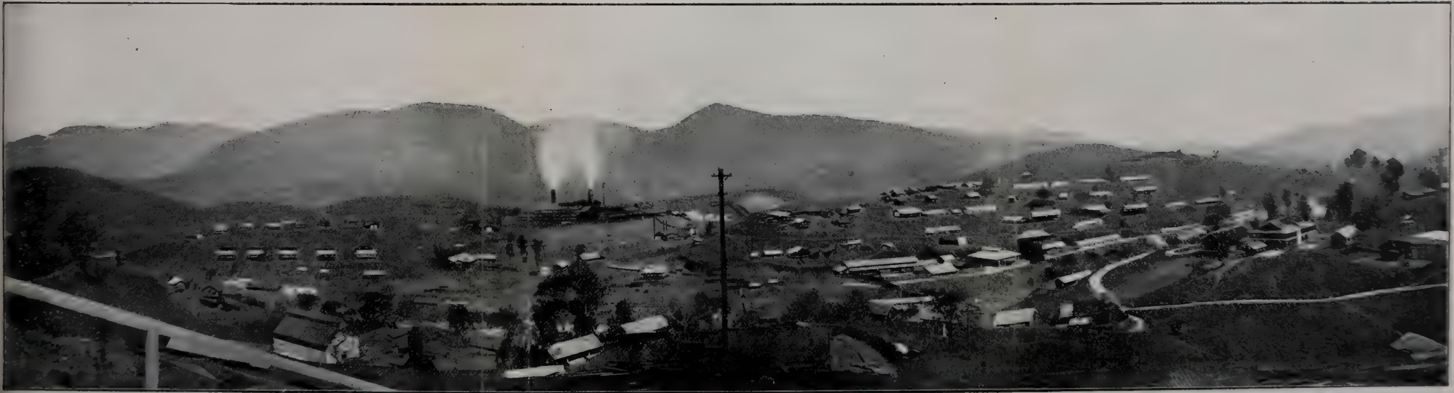


Fig. 1.
Residential Namtu.



VENTILATION AND THE KATA THERMOMETER.

BY H. H. CARROLL.

(*Read before the Broken Hill Branch, 28th February,* 1921.)

THAT atmospheric conditions have an important relation to health and efficiency has been pointed out by physiologists for a considerable time, but it is only in recent years that the practical application of this knowledge has been considered by engineers, particularly in the metal-mining industry.

As the more easily worked mines are becoming exhausted, it is found that mines are tending to become deeper, and, in consequence, new problems present themselves—not only in the methods of mining, but also in the control of conditions which affect the health and efficiency of the underground worker, and permit the carrying-on of deep-mining schemes.

Whether this latter problem is the more important in the local mines at the present time is perhaps questionable, but it is certain that, both on humanitarian and economic grounds, it deserves serious consideration.

That atmospheric conditions have an important bearing on health is generally recognized, but the effect on efficiency has not always been considered. During the recent war, however, when a maximum output per worker was required, engineers, in studying the question of efficiency, discovered that the atmospheric conditions in the workshops had an important bearing both on the health and efficiency of the worker. This fact was fully recognized by the engineers employed in designing the Ford motor-car factory. Ventilation was the first consideration, and it might well be claimed by the author of the article in the *Engineering Magazine* for November, 1914, that the factory was equipped with the best-built ventilation system ever installed in any factory building.

If ventilation affects the efficiency of the surface worker, it is quite certain that the underground worker is affected in a similar manner, and probably to a greater extent.

Many attempts have been made to fix a standard of purity of air in which it would be safe to work; but, judging by the legislation introduced and governing this factor in the mining industry, the problem has been considered from a chemical point of view chiefly, and the value of air movement has not had serious consideration.

Some of the factors affecting the purity of air are:—

(1) Percentage of oxygen present.
(2) Percentage of CO and CO_2 present.
(3) High temperatures and high humidity.
(4) Quantity supplied per employé underground.

Whilst these factors have an important bearing on the health and efficiency of the worker, their value has probably been over-estimated. We frequently find wide variations in any of the above conditions, but it does not necessarily follow that the health or efficiency is impaired on this account. For instance, we find people living and working in altitudes of 5000 ft. and over, where the oxygen content of the air is reduced owing to the lower atmospheric pressure. Or, again, in aerated water factories, where CO_2 is used in considerable quantities, the air will at times contain as much as 2 % of CO_2. No harm results in these cases.

It must not be assumed that air containing CO_2 up to 2 % is advocated, but it is necessary to point out that the system can be trained to withstand wide variations.

The effect of high temperatures and high humidity is serious, but either condition alone is bearable. Moderate temperatures up to 70° F., relative humidity 100 %, are unobjectionable, and an atmosphere nearly saturated at a temperature of 80° F. is not serious provided there is a current of air.

In the light of recent experiments conducted by Dr. Hill, it is evident that the legislation which stated that a fixed quantity must be supplied per minute was a step in the right direction. But here, again, the introduction of such a regulation did not mean that the atmospheric conditions were satisfactory. Work could be carried on in an atmosphere of 80° wet bulb, and a current of air might be passing the working place; but, unless the speed of the air current was high, the conditions could not be considered good.

From practical experience, however, we know that air movement is necessary to remove dust and fumes and humidified air from the mine, and we also find that working conditions are much more comfortable where the movement takes place.

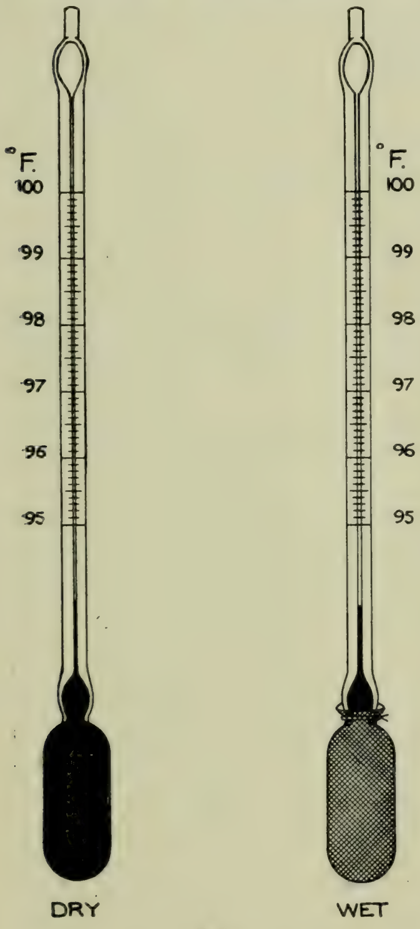


Fig 1.
The Kata Thermometer.

The value of air movement is not appreciated as it should be. We will find, however, that as the mines become deeper the factors of temperature and humidity, upon which working conditions depend at the present time, will be less easily controlled, and more reliance will be placed upon air movement.

The value of air movement has been demonstrated by Dr. Hill and his colleagues during the course of their experiments.* In one class of experiment carried out by them, several students were shut in an air-tight chamber of about 3 cub. m. capacity, which was provided with electric fans for stirring up the air, and the temperature and humidity could be varied as desired. They were kept there until the CO_2 content had reached 3 to 4 % and the oxygen had fallen to 16 or 17 %. The wet bulb temperature had risen to 83° F., and the dry a little higher. Under these conditions the degree of discomfort was considerable, but stirring up the air by means of fans gave immediate relief.

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* "The Science of Ventilation and Open Air Treatment," by Leonard Hill.

It was with the object of determining and measuring the value of this movement or cooling power that Dr. Hill conducted a series of experiments, and finally designed an instrument called the Kata thermometer (see Fig. 1) for measuring this factor of the atmosphere. Before describing the instrument, it may be advisable to mention a few physiological facts which closely affect the subject.

The body may be likened to a machine which, when it does work, produced heat and required to be kept cool. The body possesses a marvellous heat-regulating mechanism which, if not overtaxed, keeps the body at a constant temperature by :—

(1) Varying the output of moisture through the skin—*i.e.*, the heat lost by evaporation.

(2) Varying the blood circulation near the surface of the skin, or, in other words, the body, in cool surroundings with moving air, is cooled by the loss of heat by convection and radiation taking place from the skin.

Overstrain of this regulating mechanism produces serious results. We frequently find that in hot, tropical atmospheres, when a portion of this mechanism is exhausted, heat stroke occurs.

The Kata thermometer is designed to measure :—

(1) Its own rate of cooling when its temperature approaches that of the body—that is, cooling by convection and radiation.

(2) The rate of evaporation from a moist surface at body temperature.

The instrument is an alcohol thermometer having a large cylindrical bulb, with a stem graduated from 95 to 100° F. The mean of this range of temperature is approximately that of a clothed body surface, and is chosen on this account. A small bulb is provided at the end of the stem, and this gives a margin of safety in case the instrument should be overheated in preparing to take readings.

To take a dry reading, the bulb is heated in warm water till the spirit rises into the top bulb, and the column is free from bubbles. The instrument is then allowed to cool, and the warming repeated. This allows the glass time to get into equilibrium. The bulb is then dried, the instrument suspended, and the time in seconds occupied by the meniscus in falling from 100 to 95 is taken with a stop watch.

The wet bulb reading is taken in a similar manner, but before warming the instrument the bulb is covered with a muslin finger stall.

It is advisable to take several readings and average the results.

The instrument is carefully calibrated before leaving the maker's and marked on the stem with a factor. From this factor, and the time taken (in seconds) in cooling from 100 to 95, can be calculated the rate of cooling in millicalories per sq. cm. of cooling surface per second.

If H = cooling power in millicalories per sq. cm. per second.
K = factor of instrument.
T = time in secs. for meniscus to fall from 100 to 95°.

$$\text{Then } H = \frac{K}{T}$$

To enable the heat-regulating mechanism of the body to perform its duties without strain, Dr. Hill recommends the following cooling powers :—

For sedentary worker	6 dry,	18 wet.
Light manual worker	8 ,,	25 ,,
Heavy manual worker	10 ,,	35 ,,

These figures will, of course, vary, depending upon the individual and upon what is judged to be heavy work. The above figures refer to conditions obtaining on the surface. An air-current in which such readings were obtained would prevent pronounced sweating.

In many of the working places underground, at the present time, it is practically impossible to obtain readings approaching these figures.

The downcast air on the 1270-ft. level in the South mine has a temperature of 69½° F. wet bulb and 70° dry bulb. The velocity in the cross-cut, measured by releasing oil of spearmint, and at a certain point some distance along the cross-cut detecting the odour, was 35 ft. per minute. The Kata readings were 5.5 dry and 14.6 wet. Under these conditions work is performed efficiently, and excessive sweating is not apparent.

A series of Kata readings taken in the stopes on this level gave an average of 4.4 dry Kata and 11.5 wet Kata. The stopes are practically all sill floor stopes up to 150 ft. wide, and no attempt is made to direct the air current, the ventilation being by diffusion only. The men sweat freely, but, on ceasing work, cool off fairly rapidly.

The following table shows the results of readings taken in the stopes and airways throughout the mine :—

Underground Readings.

Date.	Temp.		Humidity.	Vel. in ft. per min.	Cooling Rates.		Remarks.
	Wet.	Dry.			Dry Kata.	Wet Kata.	
13/1/21— 1	70	70½	97	65	5.4	14.9	1270 level, No. 3 shaft in plat, downcast air.
2	69½	70	97	35	5.5	14.6	1270 level, No. 1 shaft in plat, downcast air.
3	71	72	94	nil	4.6	11.5	1270 level, sec. F, No. 13 party, sill floor stope ; still air.
4	70	70½	97	,,	4.8	12.4	1270 level, sec. E, No. 16 party, sill floor stope; little movement in air.
5	71	72	94	,,	3.9	10.7	1270 level, sec. A, sill floor stope ; little movement in air.
21/1/21— 6	67	68½	97	low	5.6	13.6	1170 level, No. 3 shaft in plat, downcast air.
7	68	69½	97	,,	5.7	14.1	1170 level, No. 1 shaft in plat, downcast air.
8	71	72	94	,,	4.5	12.4	1170 level, E, No. 12 party, open stope 20 ft. above sill ; air very still.
9	71½	72	97	,,	4.7	11.8	1170 level, E, No. 19 party, timber stope, 30 ft. above sill ; no movement in air.
21/1/21—10	76	77	94	,,	3.7	11.2	970 level, E., No. 47 party, large square set stope ; air still.
11	77	79	90	,,	3.2	9.6	970 level, D., No. 17 party, large square set stope, air very still, men charging holes.
12	77	79	90	,,	3.2	9.6	970 level, D., No. 50 party, large square set stope, air very still, men charging holes.
13	74½	76½	89	,,	4.8	12.4	970 level, sec. B, No. 43 party, square set stope, men using popper ; feeble current.
14	72½	76½	79	,,	4.7	13.0	970 level, sec. D, footwall No. 20, open stope, 5 mins. after firing 4 holes.

Underground Readings—continued.

Date.	Temp.		Humidity.	Vel. in ft. per min.	Cooling Rates.		Remarks.
	Wet.	Dry.			Dry Kata.	Wet Kata.	
22/1/21—15	67	70½	80½	low	5.7	17.3	970 level, No. 3 shaft in plat, downcast air.
16	68	70	88	,,	5.6	15.4	970 level, No. 1 shaft in plat, downcast air.
17	73½	74½	94½	,,	4.0	10.6	970 level, sec. H, No. 44, sill floor stope ; no movement in air.
18	73	74	94	,,	3.9	10.9	970 level, sec. H. No. 34, open stope, 50 ft. above sill ; no movement in air.
15/1/21—19	76	81	76	,,	2.7	9.6	825 level, sec. B, No. 28, square set stope, top floor ; air still.
20	75½	80½	76½	152	6.7	20.8	825 level, sec. C. No. 38, square set stope, bottom floor.
24/1/21—21	62	66	78	400	15.8	33.3	525 level, main drive, between Nos. 3 and 4 shafts.
22	65	81	39	—	4.1	23.6	425 level, in No. 1 shaft, downcast air.
14/1/21 23	60	69½	54½	60	5.6	17.3	425 level, No. 3 shaft plat, downcast air.
24	79½	83	82½	93	4.1	14.7	425 level, sec. D2, cross-cut, upcast air.
25	80	82	95	160	4.8	16.2	425 level, C2, cross-cut, upcast air from No. 2 shaft.
26	76	82½	70	207	5.0	20.0	425 level, B5, cross-cut, upcast air.
27	76½	82	74	1250	9.8	29.4	425 level, A5E, E cross-cut upcast shaft No. 6.
28	74	81½	66	250	6.2	21.6	425 level, sec. A, square set stope, top floor.

The writer would particularly draw attention to reading No. 20, which, compared with the readings obtained in the stopes on the 970-ft. level, clearly indicates the effect of air-movement. Although the temperature in this stope is 80½° F. dry bulb and 75½° wet bulb, the cooling power of the air is 6.7 dry and 20.8 wet, as against 5.4 dry and 14.9 wet at the downcast shaft.

To obtain some idea of the effect of temperature on output, a careful summary was made of the output per man from a number of square set stopes, with varying temperatures, other conditions (for all practical purposes) being the same. The following results were obtained :—

Temperature.		Velocity.	Kata Cooling Rate.		Tons per man per shift.
Wet.	Dry.		Dry.	Wet.	
70	73	15 ft. per min.	4.3	10.8	4.2
79½	80½	15 ft. ,, ,,	3.0	7.1	3.3
80	81½	15 ft. ,, ,,	2.8	6.9	3.1

These figures show a considerable falling-off in efficiency between the temperature of 73° F. dry and 80½° F. dry, and a Kata cooling rate of 4.3 and 2.8 respectively.

If we assume that the dry Kata cooling rate of 4.3 is 100 % efficient, the decrease in output amounts to 26 % when the Kata cooling rate is 2.8 dry. It cannot be definitely stated that the whole decrease is entirely due to the low cooling rates, but it is quite certain that a considerable proportion is due to this fact.

In a paper* published by Dr. A. J. Orenstein and Mr. H. J. Ireland, appearing in the *Journal of the South African Institute of Engineers* for March, 1921, the authors estimate that in a representative section of the mines 21 % of the output is lost through inadequate cooling power.

When the Kata cooling rate is 4.3 dry the men sweat freely. As sweating is an emergency method of cooling, it should be the aim of all concerned to improve ventilation and prevent, if possible, excessive sweating. It would appear that the cooling rates of 6 dry and 18 wet, as suggested by Dr. Hill for sedentary workers, while perhaps not being excellent for hard manual work, is a standard that should be aimed at for in all working places underground.

In winzes and developmental ends it is possible to obtain figures approaching the above by means of mechanical ventilation. Similar conditions also exist when a rock-drill is operating in a drive or winze. The following readings show the results obtained in a drive and a winze :—

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* "Contribution to the Study of the Influence of Mine Atmospheric Conditions on Fatigue."

Place.	Temp.		Kata.		Remarks.
	Wet.	Dry.	Wet.	Dry.	
825 level, E3e winze	74	76	19.2	8.3	Winze 58 ft. deep ; fan on.
,, ,,	76	77	12.1	3.9	Natural ventilation.
825 level, north drive	66	67½	16.2	5.6	Ordinary conditions ; low velocity.
,, ,,	63	67	20.8	8.7	Machine working, using 100 cub. ft. of free air per min.

Apart from improving working conditions in developmental ends, the supply of an additional quantity of fresh air has the effect of reducing the dust content, and this in itself is a most important consideration.

Whether we should adopt a dry or wet Kata reading as a standard does not appear to matter a great deal. It does not follow, however, that a low dry-cooling power is entirely compensated for by a high wet-cooling power.

Under these conditions, one derives some benefit, if sweating is produced, by getting the advantage of the high wet-cooling power ; but this has a disadvantage in that one is more likely to contract chills.

From the observation made, and a study of the result, the Kata thermometer appears to indicate the condition of the atmosphere and its effect on the capacity for work to a much greater degree of accuracy than the thermometer.

The Kata reading is affected by three factors — temperature, humidity, and air-movement. Provided the chemical purity of the air is about normal, working conditions are also affected by the same factors, and an instrument which indicated the value of the above conditions is of great value to all mining men engaged in the work of improving any ventilation scheme.

Whilst advocating the use of the Kata, the writer would not discard altogether the thermometer. The thermometer, however, does not always indicate the true atmospheric conditions and degree of comfort, particularly when the temperature of the air is 75° F., or higher, as reference to reading No. 20 and comparing it with reading No. 19 will show. The wet and dry bulb readings are practically the same, but the cooling power of the air in one case is two and a half times that of the other. Consequently, working conditions are better where the velocity of the air is greater.

Some experiments have been conducted with a view to using the Kata as an anemometer. It has been found, where the current of air approaches 100 ft. per min., or when eddy currents are not present, that the Kata reading can be relied upon as a means of calculating the velocity, using the formula as supplied by Dr. Hill.

When eddy currents are present, however, Kata readings cannot be checked against the readings obtained when the usual mine methods of measuring slow velocities are used. The following readings indicate this. It was possible to detect the eddy current by observing the movement of a quantity of smoke.

Place.	Temp.		Kata.		Velocity of air measured.	Velocity of air calculated.
	Wet.	Dry.	Wet.	Dry.		
1270 level, B2 crosscut	67	70	6.1	16.3	22 ft. per min.	60
1270 level, A5 crosscut	68	70	5.2	14.2	12 ft. per min.	33

For this reason direct Kata should always be used in preference to those obtained by calculation.

Dr. Hill and his colleagues have recently perfected two electrical Kata thermometers, one of which is a recording instrument.

By using these instruments the cooling powers can be obtained with very little trouble; but, as a supply of electric current is necessary, they are more likely to be used in factories, etc., than underground.

A brief description of these instruments appears in the October number of the *Bulletin of the Institution of Mining and Metallurgy* (London).

DISCUSSION.

P. H. Warren said members were indebted to Mr. Carroll for having, in his interesting and instructive paper, brought the subject of the Kata thermometer under the notice of members, some of whom were at present engaged in grappling with the local ventilation problem. The Kata thermometer was the best instrument they had for quickly measuring the cooling value of air in mines. It was not quite so readily handled as the familiar wet and dry bulb thermometer, or hygrometer, and, as Mr. Carroll pointed out, should be used in conjunction with that instrument, but enabled them to more accurately interpret the conditions of the atmosphere in underground workings, where factors such as humidity and sluggishness of air currents exercised great influence on the working efficiency of the men. On page 105 Mr. Carroll said :—"The value of air-movement is not appreciated as it should be ; we will find, however, that as the mines become deeper, the factors of temperature and humidity upon which working conditions depend at the present time will be less easily controlled, and more reliance will be placed upon air-movement." This statement brought them to a realization of the problem in ventilation which confronted them locally. The Broken Hill mines were not deep yet, and possibly some of them would never be much deeper. They could not, therefore, claim that they suffered much from the effect of high rock temperatures. So far as he knew, accurate readings of rock temperatures had never been obtained locally. The following was an attempt to estimate the geothermic gradient :—The shade temperature at the collar of No. 3 shaft, South mine, on the 14th February, 1921, at 10 a.m., was 65° F. wet, 87° F. dry, moisture content per cub. ft. of air = 3.9 gr. Temperature in No. 3 shaft, 1270-ft. level, at the same time, was 70°–71½°, moisture content 7.9 gr. water. The shaft was 1237 ft. deep to the 1270-ft. level. Allowing for heat increase due to compression at 5½° F. per 1000 ft., the temperature at 1270 ft. would be—

$$87 + 6.8 = 93.8^\circ \text{ F.}$$

As every grain of water evaporated per cub. ft. of air meant a heat loss of 8° F., they got—

$$7.9 - 3.9 = 4 \times 8 = 32^\circ \text{ F.}$$
$$93.8 - 32 = 61.8^\circ \text{ F.}$$

The actual reading on the 1270-ft. plat was 71.5° F., which left 9.7° F. to be accounted for by the influence of rock temperatures—*i.e.*,

$$\frac{1237}{9.7} = 1^\circ \text{ for } 127.5 \text{ ft.}$$

That figure could only be regarded as an approximation, for the reason that, although the air travelling through the shaft at the 1270-ft. level had little velocity, it was not still, and in all probability it was not in contact with the rock surface sufficiently long to have taken the same temperature. The mean annual temperature in Broken Hill was 64° F.; if, therefore, they took the normal temperature to be 64° at a depth of 100 ft. below the surface, and allowed 1° rise for every 127.5 ft., the rock temperature at the 1270-ft. level should be 64° + 9.7° = 73.7°. An actual reading obtained by dropping a thermometer into the water for a depth of 100 ft. below the 1270-ft. level was 76° F. The average depth of workings in the Broken Hill field was about 1200 ft., and it would appear safe to estimate the rock temperature at that point at about 75° F. Several years ago the average temperatures of the stopes on the 1270-ft. level of the South mine, under conditions of natural ventilation, were wet 70.2°, dry 70.7°, moisture content 7.9 gr. That air then ascended through the workings, and by the time the 970-ft. level was reached the average temperature of stopes in that level became 77.34° F. and 78.34° F., moisture 9.6 gr. By the time the 625-ft. level was reached temperatures were 80° and 82.5° F., moisture 10.3 gr. The air, therefore, in ascending from the 1270-ft. level sill floor stopes to the 625-ft. level stopes, had increased as follows:—Dry bulb 9.8°, wet bulb 11.8°, moisture content 2.4 gr. per cub. ft., the rock temperature at the 1270-ft. being 75°, and at the 625-ft. 69° (approximately). The question at once arose: What caused that increment in temperature and moisture content? The chief cause of the former was probably absorption of oxygen by the sulphide ore and decaying timber, while possibly also the filling may absorb oxygen with the evolution of heat, and the increase in moisture was due to absorption from rock-faces and filling. The obvious and only cure for that trouble was to impart sufficient velocity to the air, so that it would serve the working places before it had time to depreciate in value as a cooling medium. It was with the means necessary to secure that result that the mining engineer had to concern himself. They started off with excellent conditions for securing good ventilation in the mines (that is, the air was both cool and dry). The mines have not been laid out and worked in the past with any thought to ventilation efficiency, and the difficulties which had now to be met were due to the following factors :—

(1) Shafts so placed in relation to the ore-bodies that short-circuits occurred.
(2) Shafts of small sectional area in relation to the tremendous volume of the stopes to be ventilated.
(3) Openings to stopes were often inadequate in number and small in size.
(4) Openings from stopes (rises to level above) were often so small, both in size and number, as to be incapable of removing the air from the stopes in sufficient quantity to effect good ventilation.
(5) The use of timber, particularly in square-set stopes and timbered gangways, consumed most of the available motive pressure in overcoming frictional resistance.

That was rather a formidable list of difficulties to labour under, but not nearly so serious as might at first sight appear. It might be taken for granted that natural ventilation could not be relied upon to ensure a control of working conditions in respect of ventilation, for the reasons that too much of the available motive pressure was absorbed in overcoming friction, and that no control was held over either the quantity of air in circulation, or the direction of the air-currents, which varied with surface conditions. Mechanical ventilation must therefore be provided to ensure a constant supply of air. The next point was distribution. That was really a question of equalization of resistances—in other words, the resistance to the passage of the air through the stopes should be kept down to equal that of the drives, or resistance doors would have to be placed in the drives to impede the passage of the air through them. That was really the crux of the whole question, and it was due to the fact that, ordinarily speaking, insufficient openings existed through stopes between levels, that Mr. Carroll was unable to report higher Kata readings than 4 dry and 11 wet. Probably no greater reform in local mining conditions could be effected than to increase the size of openings from the stopes to the levels above. In the past it had been quite common for a comparatively large stope to be served with two winzes only, one of which—and sometimes both—would be temporarily blocked for mullocking purposes. To take a case in point, they had a stope 300 ft. long x 60 ft. wide, which was provided with 44 chutes and ladderways, by means of which air entered the stope. There were 5 winzes, 7 ft. x 5 ft. in sectional area, two of which were generally filled with mullock. That stope required 12,000 cub. ft. of air per min. to permit of an air velocity of 20 ft. per min.—a quantity altogether too great to be carried by 3 small winzes with the pressure available. The velocity through the winzes would require to be 114 ft. per min.; if, however, those winzes were stripped to carry 2 square sets each, the air velocity need only be 50 ft. per min. to carry the quantity. The Kata readings in that stope were:—Dry 4.3, wet 11.6, velocity of air current under 10 ft. per min. In order to secure a dry Kata reading of 6, the air velocity would require to be 56 ft. per min. The air velocity in the main drive leading to that stope was 147 ft. per min., with a dry cooling power of 10.5. He was quite in accord with Mr. Carroll's statement that they should aim at higher cooling rates than those which their records showed were possible at present, but he was a little doubtful if they would ever be able to get the wet cooling rate up as high as 18, owing to the fact that underground air always carried so much moisture. Readings secured recently in two of their best-ventilated stopes were 6.00 dry, 15.00 wet, and 6.05 dry, 14.40 wet, the temperature being 66° wet, 67° dry. The circulation in these stopes was particularly good.

Mr. T. Hanton said Mr. Carroll had brought before the Institute a comparatively new and at the same time a very interesting subject. The researches of Dr. Hill and his design and development of the Kata thermometer had led to the examination and consideration of working conditions from a somewhat altered aspect. The importance of air-movement as it affected comfort and working efficiency was stressed throughout by Dr. Hill. The Kata thermometer expressed in two results the heating or cooling effect of air of any temperature or humidity, and either stationary or in motion. Experience with the Kata thermometer in underground work, generally speaking, had been rather limited, and further investigation would probably be required before the value of the instrument and its sphere of usefulness in mining work could be properly assessed. In connection with a desirable Kata standard for underground working places, 6 dry and 18 wet was very commendable, but its attainment in places outside the main airways would be extremely difficult, and in many cases practically impossible. One feature of the results on the Zinc Corporation were comparatively low Kata readings in working places which experience had shown were good, and in which a high standard of efficiency had been attained. Dr. Haldane maintained that the underground worker naturally adjusted his output to his conditions. If the place was warm or humid, and there was little movement of air, he rested frequently, and so did not impair his health. His efficiency, of course, was reduced, and on that ground alone the attainment of as high a standard as possible was desirable. Mr. Carroll's figures showing the output per man were very interesting, and probably conveyed to the average mining man a clearer idea of the importance of good working conditions than did ergometer results. On fixing any Kata standard for underground work difficulty would arise on account of the variation in results over any given section of a working place. The following results were obtained on the Zinc Corporation from No. 8 level, main south drive, a downcast airway 6 ft. 10 in. high and 6 ft. 8 in. wide at the point where the readings were taken. The temperature throughout was 66° wet, 67⅓° dry. Time did not permit of the completion of the Kata figures for the remaining points on the cross-section.

Portion of Cross-section.	Anemometer Reading.	Dry Kata.	Wet Kata.	Velocity calculated from Kata Readings.	
				ft. per min.	
	ft. per min.			Dry.	Wet.
Centre of drive	166	12.3	26.5	346	475
Bottom of drive, in middle . .	62	11.1	23.2	259	287
Top of drive, E. side . .	150	9.9	22.5	184	260
Bottom of drive, W. side . .	58	6.4	17.6	39	85

The shade reading on the surface at 9 a.m. on that date was 66° wet, 88° dry; dry Kata 2.1, 3.0, and 1.5; wet Kata 33.8, 26.6, and 27.2; and wind velocity 250 ft. per min. The following readings were obtained in a stope above No. 7 level. The stope was 31 ft. wide, and the back was from 3 ft. 6 in. to 6 ft. 6 in. above the sand filling. The velocity through the stope was low; temperature was 67° wet, 69° dry.

Portion of Cross-section.	Dry Kata.	Wet Kata.	Velocity calculated from Kata Readings.	
			ft. p. min.	
			Dry.	Wet.
E. side, at top	4.4	12.3	7	15
Top, 10 ft. 4 in. from E. wall, 3 ft. 3 in. above filling, and 3 ft. 3 in. below back	5.2	14.1	19	33
Top, 20 ft. 8 in. from E. wall, 3 ft. 3 in. above filling, and 3 ft. 3 in. below back	5.1	13.3	18	24
On W. side, at top	5.5	14.1	26	33

In the light of those results the question arose as to what could be considered a representative Kata figure for any particular place. The differences were doubtless due to variations in velocity over the section, and the readings were affected by proximity to the walls. In calculating velocities from Kata readings the original formulæ of Dr. Hill were first used, but the results did not correspond with the velocities obtained by direct measurement. The velocities given above were calculated from Dr. Hill's latest formulæ, as published in the *Proceedings of the Institute of Mining and Metallurgy* of last October. As was apparent, these did not agree with the velocities measured directly. Dr. Hill stated that those formulæ were final, and they had been checked carefully at various institutions, and their accuracy borne out in each case. Underground, however, eddy currents or some other factor affected them. That eddies played a very considerable part was shown by the fact that the majority of the calculated velocities were higher than those measured directly. Another feature was that the velocities calculated from wet Kata readings were in his (Mr. Hanton's) experience invariably higher than those obtained from the corresponding dry Kata readings. Extreme accuracy was not claimed for the methods of direct measurement, but the calculated readings from the dry and wet Kata might be expected to show a closer agreement. In any event, the above figures supported the wisdom of Mr. Carroll's contention that the value of calculated Kata results was not very great. Referring to the cooling power of a fan, or of a rock-drill when working, the readings in a drive with two men shovelling, machine not working, and fan stopped, were 3.3 dry, 9.9 wet, temperature 76° wet, 76° dry. With the "Turbro" boring the readings were 6.1 dry, 17.2 wet, and the temperature 73° wet, 74° dry. The time occupied in obtaining Kata readings was, at the best, considerably greater than that necessary for temperature and velocity measurements. By the use of the electric Kata, surface readings might be facilitated, but for underground work it would, as a rule, be impracticable. The Kata thermometer would be a valuable adjunct to the equipment of the mining engineer, but whether it would entirely supplant the thermometer, and Kata readings replace the ordinary thermometer standards, was doubtful.

Mr. C. H. J. Clayton said it would interest members if he quoted an opinion given in December last to a person outside the mining profession, viz.:—"That the Kata is a useful instrument for correcting some of the anomalies incident to the use of a wet-bulb standard only, as it does take some account of cooling powers and velocities; but in the setting of a standard great care should be exercised, as by experience we have determined a low Kata reading in places where the maximum efficiency standard is attainable and there is no increased fatigue. In stopes with a temperature of 70° wet and 73° dry the Kata reading is only 4.1 and 11.2 in some of our observations, and these are admirable working places. Also, with the rock-drills working, Kata readings vary over a wide range. On the surface, at 9 a.m. on a muggy summer morning, the dry Kata reads 2.1, 3.0, and 1.5. Summing up, it would appear that if we aim for and get a dry Kata reading of 4 and a wet Kata reading of 11, it will be as much as we can reasonably expect. There are, however, so many variable factors of which we are unaware—proximity to walls, etc.—that we hesitate to name any standard until a sufficiently wide distribution of readings has been taken." He (Mr. Clayton) was of the opinion that Mr. Carroll was unduly optimistic when he aimed to get readings of 6 dry and 18 wet, and he did not think that any extra efficiency would be gained.

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By O. H. Woodward and J. C. Hughes.

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- ,, Charge Data
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SECTION A.

BLAST FURNACES—GENERAL DETAILS.

The smelting plant consists of five large furnaces operating in the treatment of current sintered material, and one small furnace operating periodically in the treatment of copper dross and antimonial slag produced in the lead-refining process.

The following table supplies the leading dimensions and details of the large furnaces :—

(*a*) *Crucible*—

Dimensions—17 ft. 2 in. long x 4 ft. wide x 2 ft. 3 in. deep.
Capacity—30 tons lead.
Class of lining—firebrick.
No. firebricks required—3000.

(*b*) *Crucible Tank*—

Dimensions—20 ft. 6 in. long x 10 ft. 6 in. wide x 4 ft. 3 in. deep.
Material—½-in. mild steel.

(*c*) *Jackets*—

Dimensions—6 ft. long x 19 in. wide x 6 in. thick.
Material—Cast-iron.
Average life—2 years.
Number per side—11, with 13-in. bosh.
,, ,, end—3, with 9-in. bosh.

(*d*) *Tuyeres*—

Diameter—5 in.
Material—Cast-iron.
Number per side—11.
Tuyere sleeves—5 ft. 3 in. long x 5 in. diam.
Diameter of bustle pipe—27 in.
Normal blast pressure—44 oz. to 54 oz. per sq. in.

(*e*) *Furnace Shaft*—

Dimensions at top of jackets—18 ft. 8 in. long x 6 ft. 2 in. wide.
Dimensions at feed floor—19 ft. 6 in. long x 7 ft. wide.
Lining—Of firebrick (8000 required).
Bricks used in construction—red brick (10,000 required).
Depth from crucible to feed floor—23 ft. 3 in.
Depth from tuyeres to feed floor—20 ft.

(*f*) *Jacket Cooling Water*—

Class of water—Salt.

Consumption per furnace—364,000 gal. per 24 hours.

(*g*) *Miscellaneous*—

Tapping breasts—Cast-iron, water-cooled, 17 in. x 12 in. x 3 in.

Diameter of tap hole—$1\frac{3}{4}$ in.

Slag spouts—Cast-iron, water-cooled.

In addition to the above tabulated data the essential sections of constructional details are dealt with more fully in the following notes.

CRUCIBLE.

The tank of the crucible has an over-all length of 20 ft. 7 in. and an over-all width of 10 ft. 6in. The four corners are cut off at an angle of 45°, so that the actual length of the end plate is 6 ft. and the side plates 16 ft.

The side of 16 ft. in length is reinforced by two 10-in. x 4-in. channel beams and 10-in. x 6-in. I-beam. On the outer flanges of these beams a $\frac{1}{2}$-in. M.S. plate is rivetted, forming two air-channels. The sketch shows the arrangement.

The ends are reinforced by three 60-lb. rails anchored back to the side plates. The body of the tank is made of $\frac{1}{2}$-in. M.S. plate, and all angles reinforced with 4-in. x 4-in. angle iron, $\frac{7}{8}$-in. rivets at 4-in. pitch. The tank is supported by a reinforced concrete foundation of a thickness of 4 ft.

Half Section Furnace Crucible (see Fig. 1).—

The dimensions of the crucible proper are :—

Length	17 ft. 2 in.
Width	4 ft.
Depth	2 ft. 3 in.

Three thousand firebricks are required in the construction of the crucible. Nairne firebricks are used, and these are of the following composition :—

	%
SiO_2	75.35
Fe_2O_3	2.30
CaO	0.50
Al_2O_3	21.60
MgO	0.18

In each crucible there are four lead syphons, two on each side of the furnace. They are in pairs directly opposite each other, and are

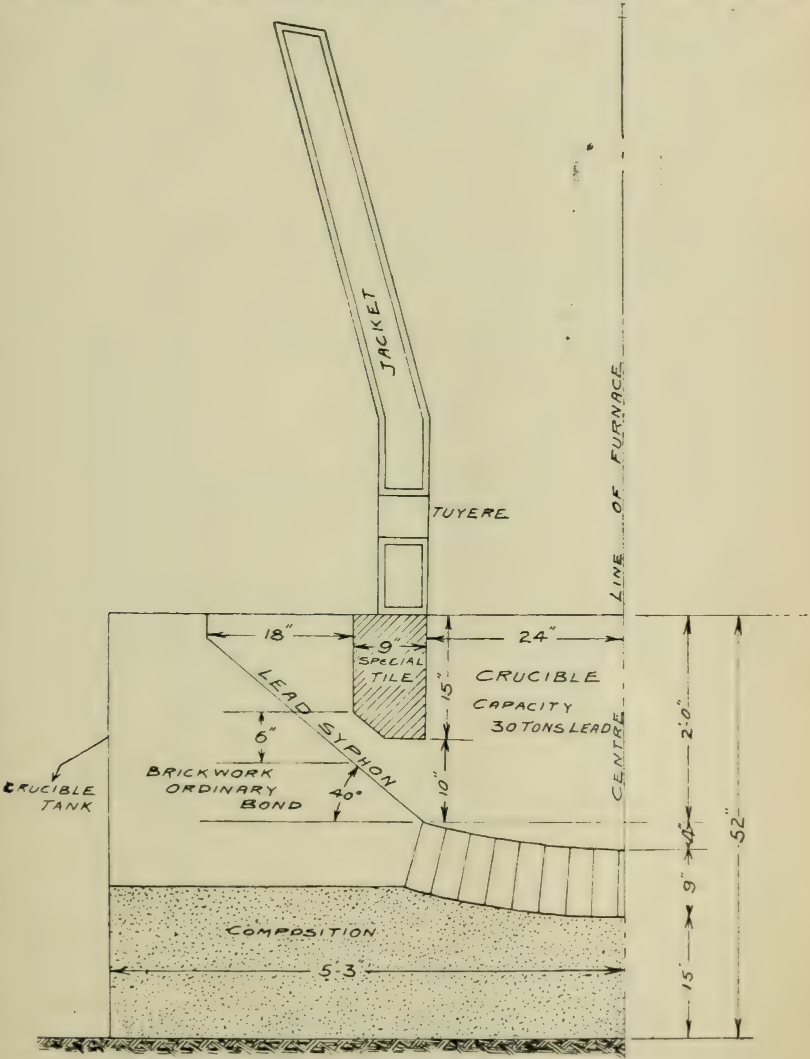
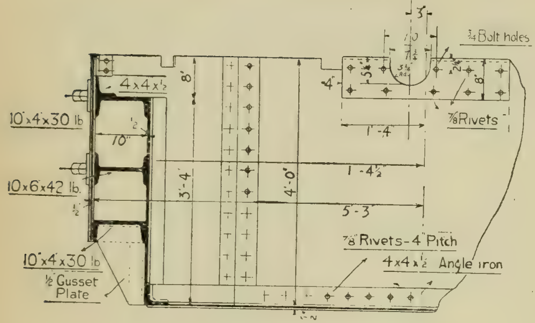


FIG. 1.

HALF SECTIONAL ELEVATION FURNACE CRUCIBLE.



Cross Section.

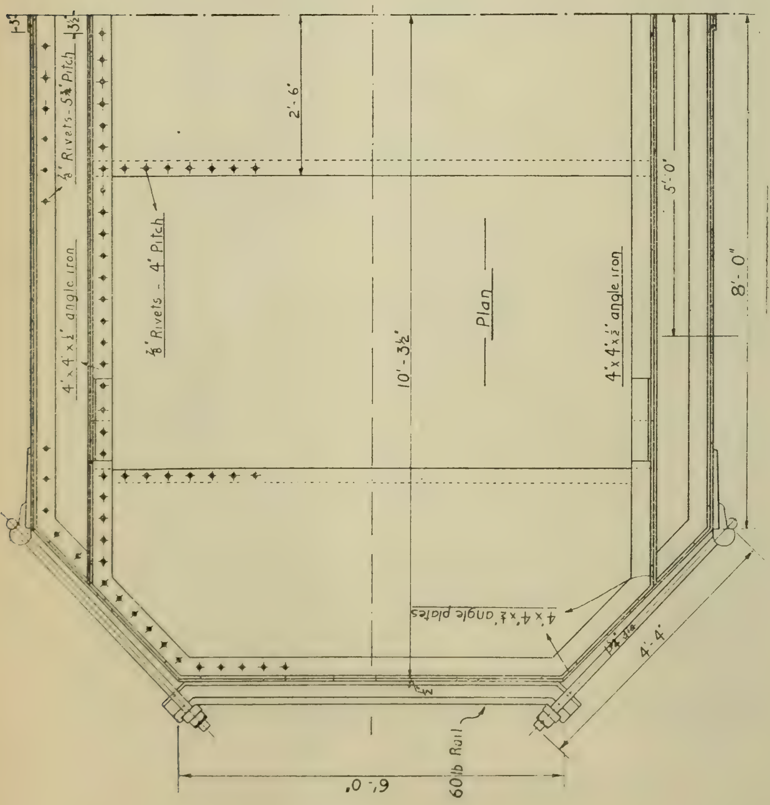
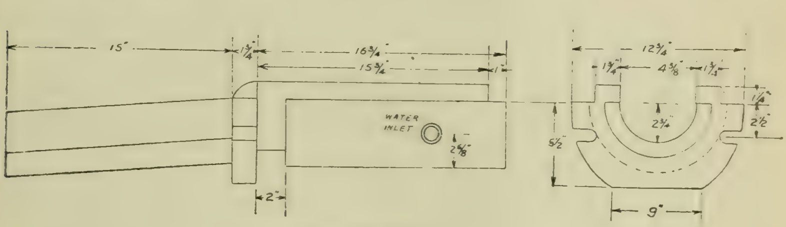


Fig. 2.
Crucible for 120-ton Smelting Furnace.

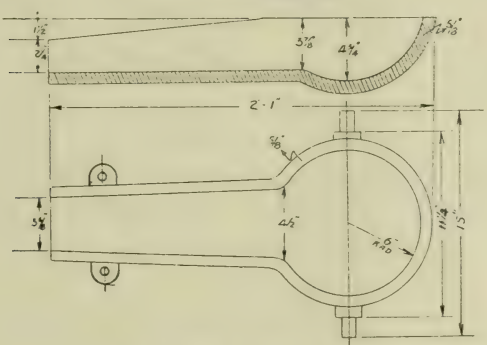
built with centres 2 ft. 8 in. from the ends of the crucible. The lead wells, or syphons, of one side only are used.

The syphon is 7 in. wide, and is surmounted on each side from the top of the crucible tank by built-up brickwork to a height of 9 in.

The lead from the syphon flows into a water-cooled lead spout, which in turn delivers to a movable spout from which the bullion moulds are filled.



SLAG SPOUT



BULLION MOULDING SPOUT

FIG. 3.

TUYERES.

Fig. 4 shows the design of the tuyere box. The junction between the tuyere pipe and the tuyere box is a simple packed gland joint. The face of the tuyere box has a shoulder which fits into a recess in the jacket casting, the whole being bolted to the jacket by three ⅞-in. studs cast into the jacket. The tuyere opening in the water-jacket is 6 in. long, and tapers from 5¼ in. diam. on the outside to 5 1/16 in. on the inner side.

The connection between the tuyere pipe and the individual tuyere coupling from the bustle pipe is made by a canvas sleeve painted inside and out with a red-lead paint and fastened to the respective members by means of screw clamps. The coupling leading from the bustle pipe is fitted with a butterfly valve, which is operated by a single weight moving in a quadrant.

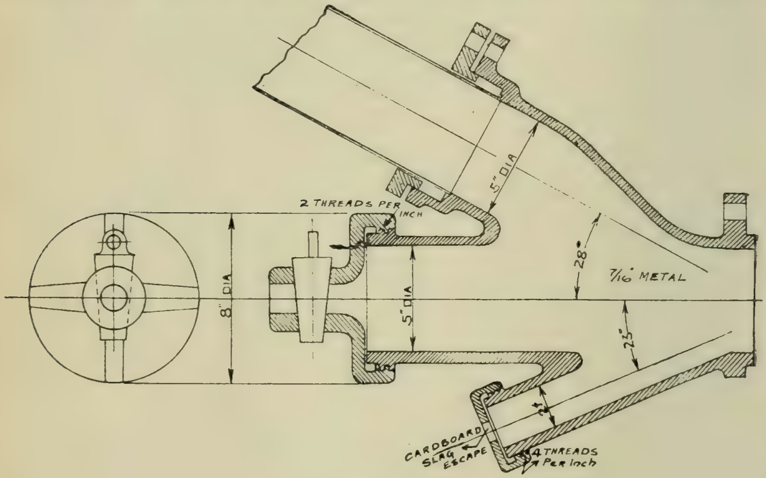


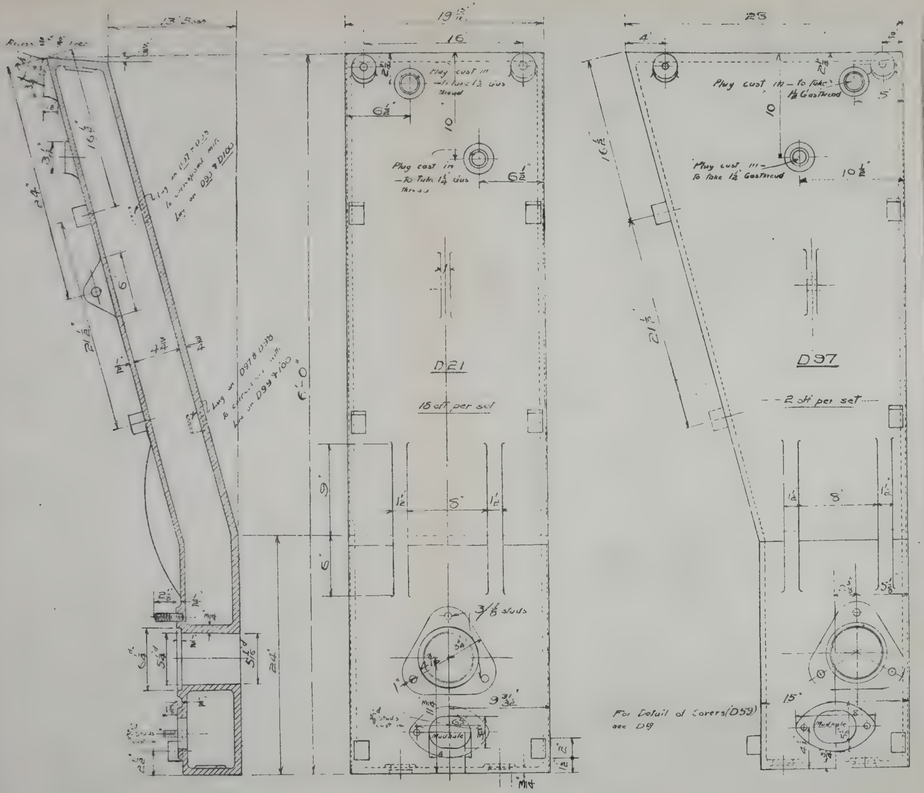
FIG. 4.
TUYERE BOX, CAST-IRON.

WATER-JACKETS.

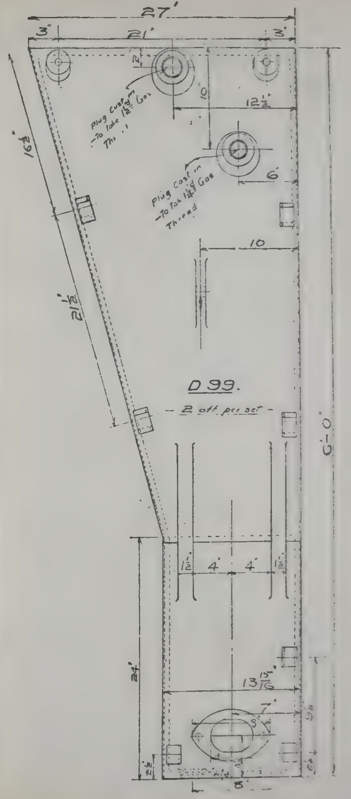
The water-jackets are of cast-iron, 6 ft. in length, and have a 13-in. bosh. The inner face of the jacket is ¾ in. thick, and the outer, or that exposed to the air, ½ in. thick, with a 4¾-in. width between walls for water space. The illustrations (Figs. 5 and 6) show the essential measurements and details of the various jackets required to complete a furnace.

The jackets required are:—

18 off pattern D21.
2 ,, ,, D97.
2 ,, ,, D97, but reversed through 180°.
2 ,, ,, D99.
2 ,, ,, D99, but reversed through 180°.
2 ,, ,, D21, but only 5 ft. 2½ in. in length over all.


13' Boss
19 13/16
16
25
4
Plug cast in — to take 1½ Gas thread
6½
10
Plug cast in — To take 1¼ Gas thread
6½
DE1
16 off per set
D97
2 off per set
Plug cast in — to take 1½ Gas thread
Plug cast in — to take 1¼ Gas thread
10½
16½
21½
24
6'-0"
Lug on D97 & D98 to correspond with Lug on D99 & D100
4¾
¾
½
9
6
1½
8
5"
3/1⅛ studs
9 31/32
Mudhole
15
For Detail of Covers (D59) see D59
2½
5¼
24

DETAILS OF FURNACE SIDE JACKETS.


27'
3"
21'
3"
10
12½
Plug Cast in — To take 1½ Gas Thread
Plug Cast in — To take 1¼ Gas Thread
6
10
16½
21½
D99.
2 off per set
6'-0"
1½
4
4
1½
24
13 15/16
7
8

END OF JACKET.

The latter, or short jackets, are in the centre of the ends of the furnace, and leave a gap in which sits the tapping breast. Fig. 7 shows the tapping breast, and it will be noted that it is 12½ in. high, whereas the height to bottom of centre jacket is 9½ in. The other 3 in. is cut out as a recess in the brickwork of the crucible.

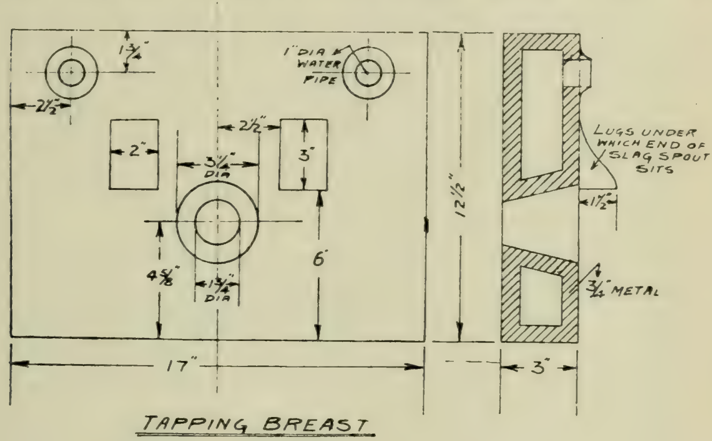


FIG. 7.

At present No. 3 furnace is the only furnace fitted with a complete set of 13-in. jackets. The other large furnaces have 13-in. bosh side jackets and 9-in. bosh end jackets. The 13-in. bosh is the adopted standard for future work.

Note.—The central jacket of the end set is, except for length, of exactly the same dimensions as the side jackets, D21. The length is 5 ft. 2½ in. over all, as against 6 ft. in D21. This gives an area of 19 15/16 in. wide x 9½ in. high, in which the water-cooled tapping breast fits. The fittings, mud-hole, studs, inlet, and outlet water couplings are the same as in D21.

BRICK SHAFT OF FURNACE.

The brick shaft of the furnace is supported on four 12-in. diam. cast-iron columns. On longitudinal section these columns are at 22 ft. 2 in. centres, and on cross section at 9 ft. 8 in. centres. Supported on these four cast-iron columns is the furnace mantle

Furnace Mantle.—The furnace mantle consists essentially of two end and two side composite girders built up of three 16 in. x 6 in. x 62 lb. I beams, and bolted together at 4-ft. centres, a cast-iron separator being fixed at these intervals. On the upper and lower flanges of this composite girder are rivetted plates 19 in. wide x 1 in. thick. The over-all dimensions of the mantle are :—Length, 24 ft. 1 in. ; width, 11 ft. 7 in. This mantle, supported on the four cast-iron columns, forms the actual support for the brick shaft.

Deck Plate.—On the top of the mantle are then placed the deck plates, details of which are given in Fig. 8.

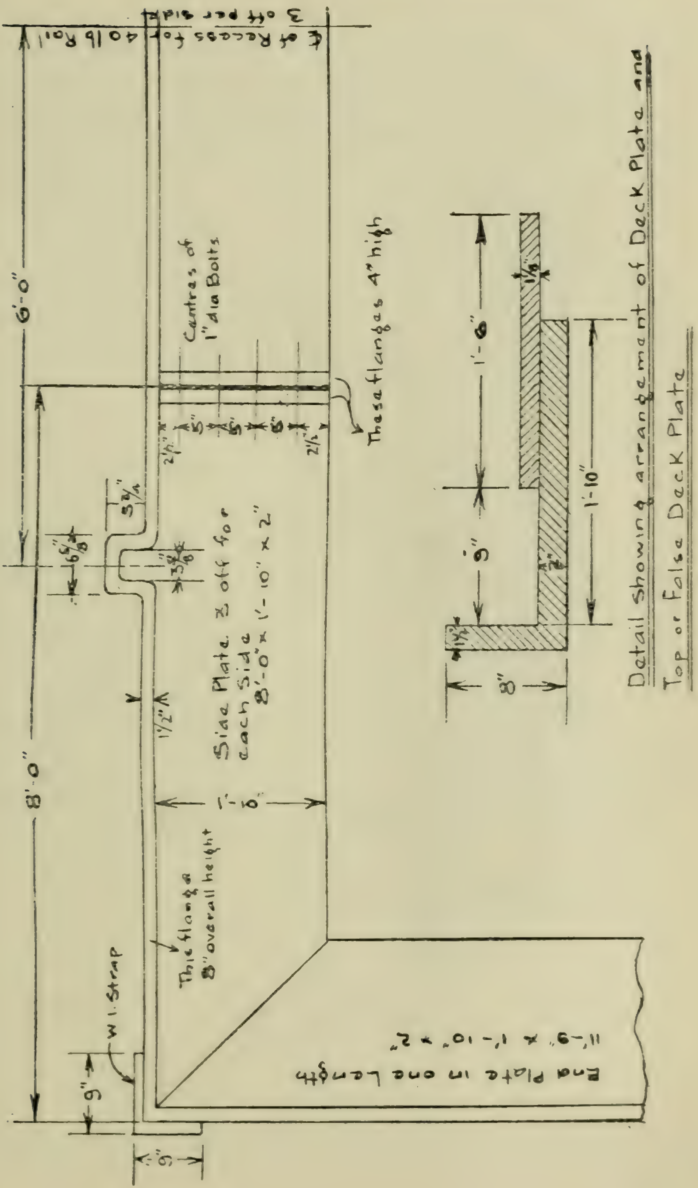
The side deck plates are 8-ft. lengths, thus giving three lengths per side of the furnace, while the end deck plates are in one length of 11 ft. 9 in. The side deck plates have a cast recess to take a railway-iron buckstay. These recesses are cast in each plate so as to give 6-ft. centres. There are no buckstays on the furnace ends, consequently the end deck plates are plain castings. On the main deck plate a top or false deck plate is fixed in a position as shown in sketch. The object of this is to give a sufficient foundation to the walls of the shaft and also to afford an easily-replaceable section of the deck plate system should the plate be burnt out.

Connection between Deck Plate and Water-Jackets.—The connection between the top of the water-jackets and the top or false deck plate is made by a special block from B. H. A. S. block mixture. The illustration shows details of this block, the top of which carries a recess 5 in. long by 1½ in. deep to support the outer end of the false deck plate, and the bottom has a long corner to conform with the top of the water-jacket. The vertical joint between the blocks is broken by a scarfed joint.

Details of Shaft Proper.—The following are the leading dimensions of the brick shaft :—

At top of jackets—length, 18 ft. 8 in. ; width, 6 ft. 2 in.
At feed floor—length, 19 ft. 6 in. ; width, 7 ft.
Height of shaft from top of jackets to feed floor—15 ft.
Batter of side and end walls—1 in. in 3 ft.

Above the deck plate for a height of 5 ft. 9 in. the thickness of the brickwork is—at deck plate, 2 ft. 9 in. ; at height of 5 ft. 9 in., 2 ft. 7 in. The inner section of the wall is 13½ in. thick, and is built of good quality fireclay brick, usually Bulli brick. The outer section is built of good quality red brick.


6'-0"
8'-0"
℄ of Recess for 40 lb Rail
3 off per side
Centres of 1" dia Bolts
These flanges 4" high
2½"
5"
5"
5"
2½"
3¾"
6⅝"
3⅝"
Side Plate 2 off for each Side 8'-0" × 1'-10" × 2"
1½"
1'-5"
This flange 8" overall height
W.I. Strap
9"
9"
End Plate in one Length 11'-9" × 1'-10" × 2"
1'-6"
1⅛"
1'-10"
9"
2"
1½"
8"

Detail showing arrangement of Deck Plate and Top or False Deck Plate

FIG. 8.

At the height of 5 ft. 9 in. the outer or ordinary brick section of the wall is given a 9-in. inset so that the total thickness of the wall from this point to the feed floor is 22 in., less the batter on the inner face. The inner wall of this section is built for the first 3 ft. in height of Nairne firebricks, and the top section of the first quality red brick.

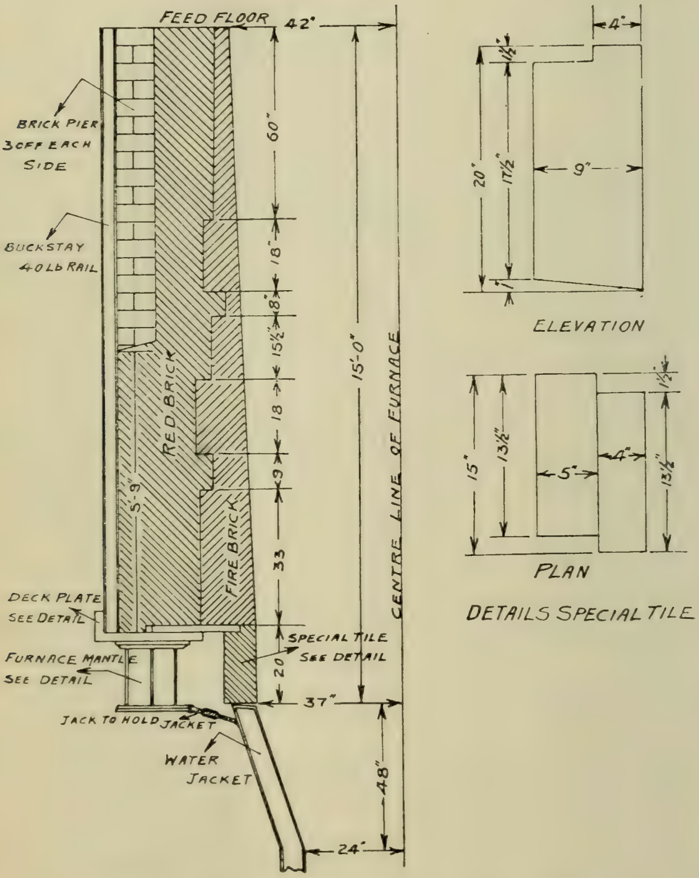


Fig. 9.
Furnace Shaft—Half Sectional Elevation.

Fig. 9 shows the general arrangement and method of bonding the two classes of brickwork.

Method of Staying Brickwork.—In vertical elevation (Fig. 9) above the deck plate 4-in. x 4-in. x $\frac{1}{2}$-in. T irons are let into the outer face of the brickwork. There are five of these T irons, the first being 15 in. above the deck plate and the others at 2-ft. centres.

On each side of the furnace are three railway-iron buckstays set into a recess in the deck plate and held on to the top by $1\frac{1}{4}$-in. diam. tie rods. Above the point where a 9-in. inset of the outer wall takes place, a buttress is built out to meet these railway-iron buckstays.

The lower and upper rectangular sections of the shaft are stayed by 5-in. x 5-in. x $\frac{3}{4}$-in. angle irons on the corners, fitted with special junction for tie rods. In vertical section there are three tie rods in the lower section and three in the upper, the former being 1-in. diam. and the latter $\frac{7}{8}$-in. diam. These rods are equally spaced in each section.

FURNACE DOWNTAKES, MAIN FLUE, STACK.

Downtakes.—On the 120-ton furnaces the central flue, which has an inverted V-shaped top, dips into the shaft 1 ft. below the feed floor level and rises 6 ft. 6 in. above feed floor level. This flue extends the whole length of the furnace, and is 3 ft. in width. From each end of the central flue a downtake, 4 ft. x 4 ft., takes the furnace gases to the main flue. From the longitudinal centre line of the furnaces to the longitudinal centre line of the main flue the distance is 31 ft.

The bottom of the down-comer enters the main flue 12 ft. above ground level, so that in this distance the downtake drops 12 ft. 9 in.

The downtakes are at 36-ft. centres.

Each downtake is 4 ft. x 4 ft. in cross section, and is made of $\frac{1}{8}$-in. plate and $2\frac{1}{4}$-in. x $2\frac{1}{4}$-in. angle iron.

(*b*) *Main Flue.*—This flue is constructed of ordinary red brick with side walls 2 ft. 6 in. thick, surmounted by an arch of 9 in. thickness. The spring of the arch is 10 ft. from floor level of flue. The arch is struck to a radius of 7 ft. 6 in., the centre being 9 ft. 6 in. above floor level of the flue. The internal dimensions of the flue are—width, 15 ft.; height to centre of arch, 17 ft.; and length, 520 ft. On the sides are put holes for removing the settled dust. One man is employed removing the accumulated flue-dust.

(*c*) *Stack.*—This is a self-supporting steel stack, 205 ft. total height and 14 ft. internal diameter. The stack foundation consists of concrete, 36 ft. sq. and 6 ft. deep, resting on 30-ft. pile work (on account of poor natural foundation). On this foundation is erected a masonry base 25 ft. high and 28 ft. over-all diameter. The walls are 7 ft. thick, leaving an internal diameter of 14 ft., agreeing with the diameter of the stack proper. The connection between main flue and stack is 17 ft. high x 9 ft. wide.

The steel section of the stack is coned for a height of 36 ft. above the top of the masonry base, having at the bottom a diameter of 21 ft. 6 in., and at the top a diameter of 15 ft. 6 in. From the top of the cone to the top of the stack—*i.e.*, 144 ft.—the diameter of the steel shell is 15 ft. 6 in. The steel shell is lined, and the lining, for various heights above the masonry base, has the following thickness:—

ft.	ft.			ft.	in.
0—	6 ;	thickness of	lining,	3	6
6—	11	,,	,,	2	0
11—	16	,,	,,	1	6
16—	28	,,	,,	1	2
28—	130	,,	,,	0	9
130—	180	,,	,,	0	4½

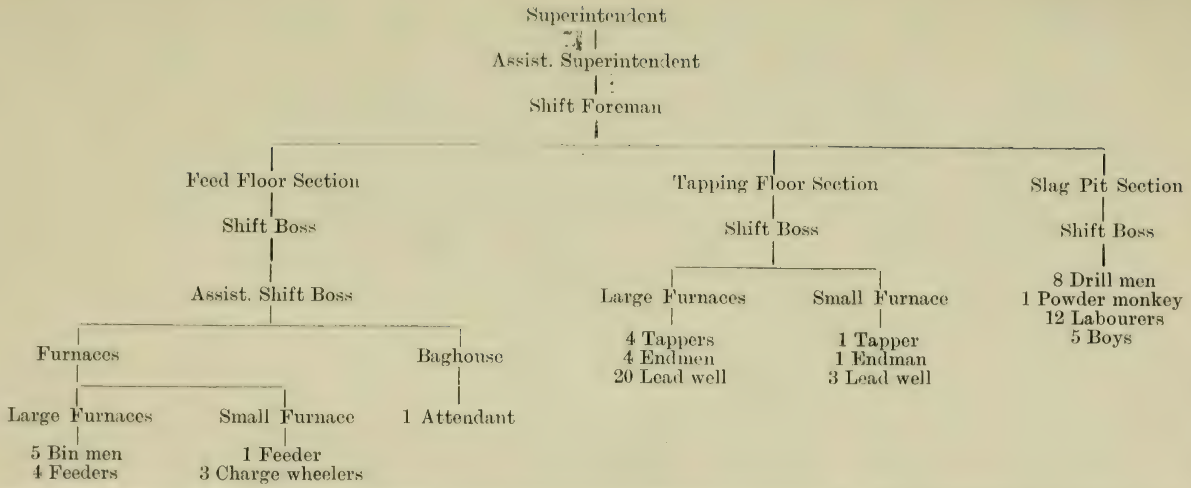
SECTION B.

(*a*) DEPARTMENTAL ORGANIZATION.

The accompanying flow-sheet shows the organization of the blast furnace department. This department is controlled by a superintendent, with an assistant. Each shift is controlled by a shift foreman. The flow-sheet gives the total allocation of all labour employed in the department.

Although the labour flow-sheet gives the various grades of employment on the furnaces, it is to be noted that no line of demarcation exists in the nature of the work on which the various sections may be employed should the furnace conditions require all the labour being employed at one time on one specific job.

Thus, for example, when a furnace is being barred, all labour which can be spared from the tapping floor is sent to the feed floor to assist in the operation. However, in such cases it is the practice for those labourers who daily receive a special rate to remain on that rate, no matter what the nature of their employment. This applies particularly to feeders and tappers.



NOTE.—The bin men also attend to the preparation of sinter and slag and coke of charge to small furnace. 3 charge wheelers are required on small furnace as Cu. dross, scrap iron and antimonial slag are stored in bins on ground level. Slag pit works day shift only. Part of labour picking matte from No. 7 furnace.

GENERAL LABOUR.—2 men on slag disposal. †1 man on recovery of flue dust from main flue. 1 man greaser. † Cribhouse attendant. † Rouseabout.

Labour requirements based on 4 large and 1 small furnace operating. All labour other than slag pit and general labour marked thus † shown on shift requirements.

It will be noticed in the text of this article that reference is made to the treatment of refinery by-products in the small blast furnace. The treatment of these refinery by-products is a debit to the refinery department and a credit to the blast furnace department. The refinery department is debited at a rate per ton equal to the cost of normal smelting, as shown by the blast furnace current monthly costs.

(*b*) FURNACE RECORDS.

The following report forms are self-explanatory regarding the records of tuyeres, blast pressure, and bag-house operations.

THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.,
PORT PIRIE, S.A.

Port Pirie,..............................192......

TUYERE RECORD.

No. of Furnace.	8 to 4.		4 to 12.		12 to 8.	
	Bad Tuyeres at Start.	Bad Tuyeres at Finish.	Bad Tuyeres at Start.	Bad Tuyeres at Finish.	Bad Tuyeres at Start.	Bad Tuyeres at Finish.
1						
2						
3						
4						
5						
6						
7						

REMARKS :—..

....

Foreman—8 to 4. *Foreman—4 to 12.* *Foreman—12 to 8.*

THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

FURNACE RECORDS. BLAST RECORD.

24 Hours Ending 8 p.m....*19*

NO. OF FURNACE.	9 a.m.	10 a.m.	11 a.m.	12 noon.	1 p.m.	2 p m.	3 p.m.	4 p.m.	5 p.m	6 p.m.	7 p.m	8 p.m.	9 p.m	10 p.m.	11 p.m.	12 noon.	1 a.m.	2 a.m	3 a m.	4 a.m.	5 a.m.	6 a.m.	7 a.m.	8 a.m.
I.																								
II.																								
III.																								
IV.																								
5.																								
6.																								
9.																								
REMARKS.																								

Foreman.	Foreman.	Foreman.

THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD., PORT PIRIE.

PLANT OPERATIONS. COMPARISON OF METALLURGICAL RESULTS.

Period ending / /192 with Period ending / /192 .

BLAST FURNACE DEPARTMENT.

Reference No.	ITEM.	Period from to	Period to Date.	Period from to	Period from to
1	Furnace Working				
2	Total Furnace Hours worked				
3	Less Stoppages				
4	H. & H. Material—Tons				
5	% Lead				
6	Ozs. Ag				
7	% S.				
8	D. & L. Material—Tons				
9	% Lead				
10	Ozs. Ag				
11	% S.				
12	Miscellaneous Parcels—Tons				
13	Tons of Lead				
14	Limestone in Sinter				
15	Limestone to Furnaces				
16	% on Charge				
17	% on New Input Lead				
18	Ironstone in Sinter				
19	Ironstone to Furnaces				
20	% on Charge				
21	% on New Input Lead				
22	Silicious Ore in Sinter				
23	Silicious Ore to Furnaces				
24	% on Charge				
25	% on New Input Lead				
26	Scrap Iron				
27	% on Charge				
28	% on New Input Lead				
29	Total Charge—Tons				
30	Total New Input Lead—Tons				
31	Copper Dross—Tons				
32	Lead Dross—Tons				
33	Sundries—Tons				
34	Retd. Slag in Sinter				
35	Retd. Slag in Furnaces				
36	% on Charge				
37	% on New Input Lead				
38	Fuel, Coke—Tons				
39	% on Charge				
40	% on New Input Lead				
41	Total Slag made—Tons				
42	% Lead				
43	Ozs. Ag				
44	% ZnO				
45	Tons Slag per Ton Input Lead				
46	Tons New Slag per Ton Input Lead				
47	Slag to Dump—Tons				
48	% Lead				
49	Slag to Pit—Tons				
50	% Lead				
51	Slag to D. & L.—Tons				
52	% Lead				

SMELTER BAG HOUSE RECORD.

24 hours ending 8 a.m. 192

Time.	PRESSURE.		Temp. F.	REMARKS.
	Before	After		

.... *Shift Boss.*

SECTION C.

BLAST FURNACE METALLURGICAL RECORDS.

The information necessary to prepare these records is obtained from the blast furnace charge sheets (see Sheet A). A charge sheet is supplied for each furnace in operation, and on it is entered the pounds of each material in the charge and also the number of charges fed.

Summarizing the sheets for all furnaces, the total gross tonnage of each material fed to the blast furnaces is obtained. To these figures is added 5 % in order to obviate scale errors. This, plus error on the weighbridge operation, has been found to be around 5 % over long periods.

In order to obtain the net tonnage of sinter treated, a moisture of 2.7 % and 0.5 % respectively is deducted from the * Huntington & Heberlein and † Dwight & Lloyd sinters.

---

* Hereinafter referred to as H. & H.

† Hereinafter referred to as D. & L.

The tonnage of each material treated daily is then posted to a weekly summary sheet (see Sheet B), and weekly totals obtained.

The bullion production is also entered on this sheet daily. This information is obtained from the number of bars of bullion produced as per charge sheets and the bullion contractor's tally book.

The weight factor for obtaining the tonnage from the number of bars of bullion is obtained from the average weight of a truck of bullion, weighed before and after crib on each shift. The trucks carry a standard number of bars, and the weighing is checked by the refinery shift foreman.

The slag production and disposal, as shown on the blast furnace daily memo., is entered on Sheet C. The assays are entered daily, and the contents worked out.

Sheet B supplies all the data referable to one week's operation, and shows the total of each item for the week. The total of the weekly figures is then transferred to a form (Sheet D), but which gives in greater detail the assay-values and contents of the materials treated. This sheet takes care of the weekly totals, so that, at the end of each costing period, the totals are readily ascertained. Also, the staff is able to arrive at the tonnages treated and/or produced for half-year to date by bringing forward the totals from previous periods.

The bullion production shown on this return is allocated between bullion produced from ore and bullion produced from drosses. This is arrived at by applying to the new lead and silver input a bullion and silver recovery equal to that obtained in the previous half-year. The difference between these figures and the total figures referable to lead and silver is considered as being respectively produced from drosses. This return (Sheet D) is then supplied to the cost office, and also supplies the data upon which the four-weekly metallurgical returns are compiled.

DAILY FURNACE RECORD SHEET.

On Sheet E is recorded the daily furnace records, slag analysis, and slag disposal in a manner allowing ready reference for the blast furnace superintendent.

THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD., PORT PIRIE. Form D/5

LEAD SMELTERY.

O –2453. *No....................Blast Furnace Charge Sheet 24 hours ended....................19*

Material.	Number of Charges Fed.																					TOTALS.			
	DAY SHIFT.							AFTERNOON SHIFT.							NIGHT SHIFT.							(To be filled in at Office).			
																						Gross Weight.		H₂O.	Dry Weight
																						Pounds.	Tons.		Tons.
Dwight Lloyd Mixture H. & H. Mixture																									
Total Refinery Drosses																									
Copper Dross Lead Dross Gold Litharge Silver Litharge Gold Alloy Slag																									
Bag House Material																									
Ironstone Scrap Iron Limestone Silica Rock																									
Slag																									
Total Charge																									
Coke																									

Bars of Bullion Made
Day Shift from Fees
Aft. ,, ,, ,
Night ,, ,, ,,
,, Kettle for all Furnaces for 24 Hours (Enter on No. 1 Furnace Sheet only).

Total Charges Fed
Day Shift
Aft. ,,
Night ,,

Pots of Slag Made (weighing tons each).
Day Shift
Aft. ,,
Night ,,

Stoppages.
Day Shift Hours for
,, ,, ,, ,,
,, ,, ,, ,,
Aft. ,, ,, ..
,, ,, ,, ,,
,, ,, ,, ,,
Night ,, ,, ,,
,, ,, ,, ,,
,, ,, ,, ,,

Day Shift Boss.
Afternoon Shift Boss.
Night Shift Boss.
Superintendent.

SHEET A.

THE BROKEN HILL ASSOCIATED SMELTERS

LEAD

BLAST FURNACES.

Date.	Furnace No.	Time—Hours.	SINTER.		SUNDRIES.						TOTAL
			H. & H. Tons.	D. & L. Tons.	Cu. Dross. Tons.	Pb. Dross. Tons.	Au. Litharge. Tons.	Au. Alloy. Slag Tons.	Bag House. Product Tons.		Tons.

REMARKS—

......................................Smelter Supt.

BROKEN HILL ASSOCIATED SMELTERS

Slag Made Week ended

Date.	Slag to D. & L.					Slag to Dump.					Slag From No. 7.				
	Tons.	Assay.		Contents.		Tons.	Assay.		Contents.		Tons.	Assay.		Contents.	
		% Pb.	Ag. ozs.	Pb.	Ag. ozs.		% Pb.	Ag. ozs.	Pb.	Ag. ozs.		% Pb.	Ag. ozs.	Pb.	Ag. ozs.

PROPRIETARY LTD., PORT PIRIE. Form D/1

SMELTERY.

Week ended .... 19......

FLUXES, ETC.					TOTAL.	RETURNS.		FUEL.		BULLION UNDROSSED.			SLAG. MADE	
Limestone. Tons.	Ironstone. Tons.	Silica Rock. Tons.	Siliceous Ore. Tons.	Scrap Iron. Tons.	Charge.	Returned Slag. Tons.	Returned Matte. Tons.	Coke. Tons.	% on Total Charge.	Gross Tons.	Ag. Ozs.	Au. Ozs.	Tons.	% Lead.

....Plant Supt.

SHEET B.

PROPRIETARY LTD., PORT PIRIE.

....192

SLAG TO PIT.					SAVED SLAG.					TOTAL SLAG.				
Tons.	Assay.		Contents.		Tons.	Assay.		Contents.		Tons.	Contents.			
	% Pb.	Ag. ozs.	Pb.	Ag. ozs.		% Pb.	Ag. ozs.	Pb.	Ag. ozs.		Lead.	Silver.		

SHEET C.

THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD., PORT PIRIE.

Form D/5

LEAD SMELTERY.

O—2199 BLAST FURNACES. Statement for Week Ended........................19

	CURRENT WEEK.							CURRENT HALF-YEAR TO DATE.				LAST HALF-YEAR.			
		ASSAYS.			CONTENTS.				CONTENTS.				CONTENTS.		
	Dry. Tons.	% Pb.	Ozs. Ag.	Ozs. Au.	Lead, Copper. and/or Dross. Tons.	Sil-ver. Ozs.	Gold. Ozs.	Tons.	Lead. Copper, and/or Dross Tons.	Sil-ver. Ozs.	Gold. Ozs.	Tons.	Lead, Copper, and/or Dross. Tons.	Sil-ver. Ozs.	Gold. Ozs.
TREATMENT.															
ORE MIXTURE—															
H. & H. Mixture															
"Cross Over" Mixture															
D. & L. Mixture															
TOTAL ORE MIXTURE															
REFINERY DROSSES—															
Copper Dross															
Leady Drosses															
Gold Pressed Dross															
Litharge—Ordinary															
Litharge—Silver															
Gold Alloy Slag															
Cupel Bottoms															
Gold Litharge															
TOTAL DROSSES															
GRAND TOTAL METAL BEARING MATERIALS TREATED (Excluding Smelter By-Products)															

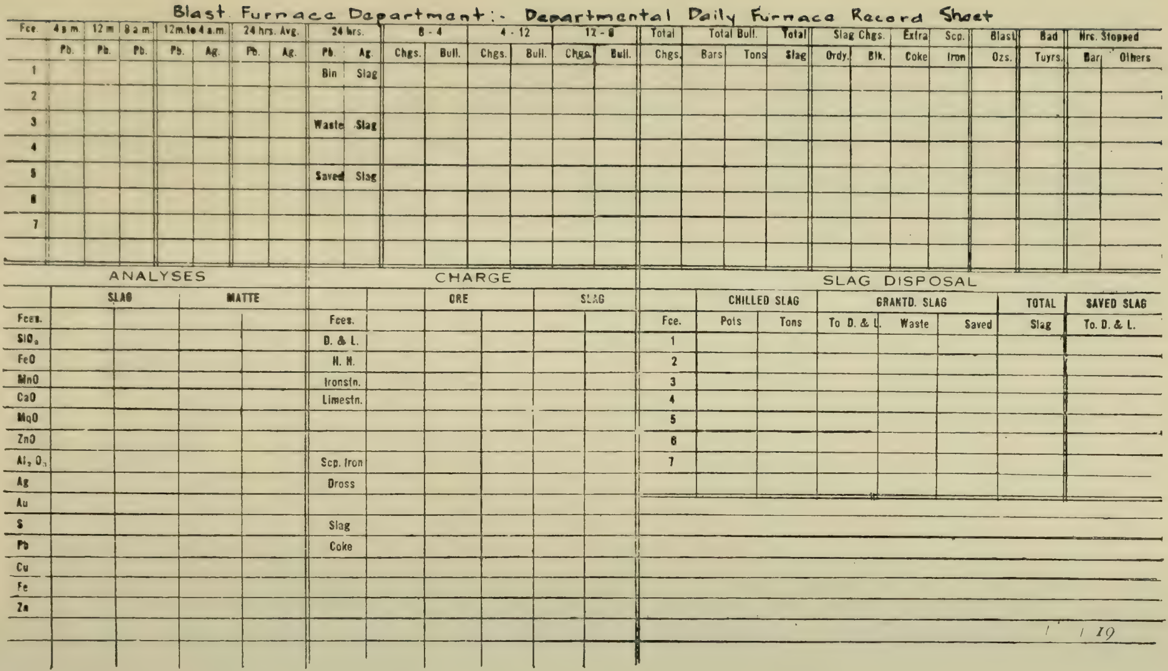
PRODUCTION.															
Bullion on Hand previous Week Brought Forward															
Bullion produced from Ore (See Form E/8															
Bullion Produced from Refinery Drosses															
TOTAL BULLION															
Bullion Delivered to Refinery															
From Ore (See Form E/8)															
From Drosses															
Balance on Hand at Smelters															
TOTAL BULLION AS ABOVE															
RECOVERIES:—Lead %															
Silver %															
Gold %															
SMELT. BY-PRODUCTS RE-TREATED															
Bag House Material															
FLUXES AND FUEL USED.															
Limestone															
Limeshel.															
Silica Rock															
Ironstone															
Scrap Iron															
Slag															
Coke															
SLAG MADE.															
FURNACES RUNNING:—No 1	Days							Dys				Dys			
2															
3															
4															
5															
6															
9															

Port Pirie, 19

Smelter Superintendent.

General Manager.

SHEET D.



Blast Furnace Department:- Departmental Daily Furnace Record Sheet

Fce.	4 a.m.	12 m	8 a.m.	12m.to 4 a.m.		24 hrs. Avg.		24 hrs.		8 - 4		4 - 12		12 - 8		Total	Total Bull.		Total	Slag Chgs.		Extra	Scp.	Blast	Bad	Hrs. Stopped	
	Pb.	Pb.	Pb.	Pb.	Ag.	Pb.	Ag.	Pb.	Ag.	Chgs.	Bull.	Chgs.	Bull.	Chgs.	Bull.	Chgs.	Bars	Tons	Slag	Ordy.	Blk.	Coke	Iron	Ozs.	Tuyrs.	Bar	Others
1								Bin	Slag																		
2																											
3								Waste	Slag																		
4																											
5								Saved	Slag																		
6																											
7																											

ANALYSES					CHARGE					
	SLAG		MATTE				ORE		SLAG	
Fces.					Fces.					
SiO_2					D. & L.					
FeO					H. H.					
MnO					Ironstn.					
CaO					Limestn.					
MgO										
ZnO										
Al_2O_3					Scp. Iron					
Ag					Dross					
Au										
S					Slag					
Pb					Coke					
Cu										
Fe										
Zn										

SLAG DISPOSAL

	CHILLED SLAG		GRANTD. SLAG			TOTAL	SAVED SLAG
Fce.	Pots	Tons	To D. & L.	Waste	Saved	Slag	To. D. & L.
1							
2							
3							
4							
5							
6							
7							

/ / 19

SHEET E.

SECTION D.

ROUTINE SAMPLING AND ASSAYS.

The following notes indicate the routine sampling and assay determinations in connection with the blast furnace department. It is to be noted that all ores, fluxes, etc., are of known analysis prior to being delivered to the blast furnace department, and the following notes apply to furnace routine only.

1. *Slag ex Furnaces.* — Samples are taken from each furnace operating at 8 a.m., 4 p.m., and 12 midnight, and a shift determination made for lead only. The sample is taken from the chute of the elevator delivering into the slag storage bin.

2. *Slag : Metallurgical Analysis.*—A sample is taken at half-hourly intervals from the chute of each elevator, and a composite sample made from all furnaces operating. A daily analysis is made for SiO_2, FeO, MnO, CaO, and ZnO. A portion of the above composite sample is set aside daily, and an aliquot part sample for a fortnight made up and asayed for Al_2O_3, MgO, Cu, and S. An aliquot part sample from the slag produced by each furnace is taken separately and the Pb and Ag determined, giving a 24-hour average sample from each furnace.

3. *Granulated Slag Delivered to Dump.*—A sample is taken by means of a spear from each truck of slag sent to the dump. The total sample thus taken is thoroughly mixed, and from this sample a daily Pb and Ag determination is made, giving the metal values lost in slag. Again, a portion of this daily sample is set aside, and from it at the end of each half-year an aliquot part composite sample is prepared. On this sample a complete analysis is made, giving the average composition of the slag produced over half-yearly periods.

4. *Returned Slag ex Quarry.*—A daily sample is taken from the feed trucks on the blast furnace feed floor, and a determination made for Pb and Ag.

5. *Bullion.*—By means of a bullet-mould a sample is taken from the running stream for every 40 bars of bullion moulded. From this daily sample a composite weekly sample is made up and assayed for Ag, Au, Cu, As, and Sb.

6. *Bag-house Material.*—A sample from every truck, after sintering, is taken daily, and from this a composite half-yearly sample is prepared. On this composite sample a half-yearly complete analysis is made.

7. *Matte ex No. 7 Furnace.*—A daily sample is taken from the slag pot while at the furnace by means of a ladle. It is assayed for Pb, Cu, and Fe.

SECTION E.

DELIVERY OF ORES, FLUXES, COKE, ETC., TO SMELTER BINS.

The following constitute the materials which are delivered direct to the blast furnace charge bins :—

Class of Material.	Delivered from.
D. & L. sinter	D. & L. roasting department.
H. &. H sinter	Crossover roast department.
Refining furnace dross	Refinery department.
Gold alloy furnace slag	Refinery department.
Silicious ore	Purchased ore stocks.
Limestone	Stock pile.
Ironstone	Stock pile.
Returned slag	Slag pit quarry.
Coke	Stock pile.

The system of bins to accommodate this material consists of four large bins and a fifth bin of equal total tonnage capacity, divided into two sections. The four large bins have a volume of approximately 10,560 cub. ft , and hold about 700 tons of material. These bins carry D. & L. sinter, H. & H. sinter, returned slag, and coke, but in the case of the latter bin the storage capacity is only 350 tons of coke. The fifth bin is divided into two sections, with a storage capacity of about 350 tons. One section is reserved for refinery department by-products, such as refiner drosses and gold alloy slag ; the other section is reserved for the storage of mixed silicious ore, limestone, and ironstone, these materials, individually and collectively, being delivered to the bin in the correct proportions as required for the furnace charge.

In addition to the above materials delivered to the bins, other refinery by-products—viz., copper dross and antimonial slag—together with scrap iron, are delivered to the smelters, and are stored in paddock bins on ground level and delivered to the feed floor by means of hand-drawn charge carts and hydraulic lifts. These latter materials are treated in the small furnace, No. 7.

DETAILS OF BIN CONSTRUCTION.

The bin proper is supported upon an understructure of framed 10-in. x 10-in. oregon timber, the base of which is 26 ft. 10 in. x 24 ft. 10 in. over all. The height of the understructure from ground level is 38 ft. 8 in. Lengthwise on the understructure are five 10-in. x 10-in. oregon studs, while in width there are seven 10-in. x 10-in. oregon studs, making five rows of 7 studs. On top of each row of studs are placed two caps of 10-in. x 10-in. oregon, one superimposed on the other. This forms the foundation of the bin proper.

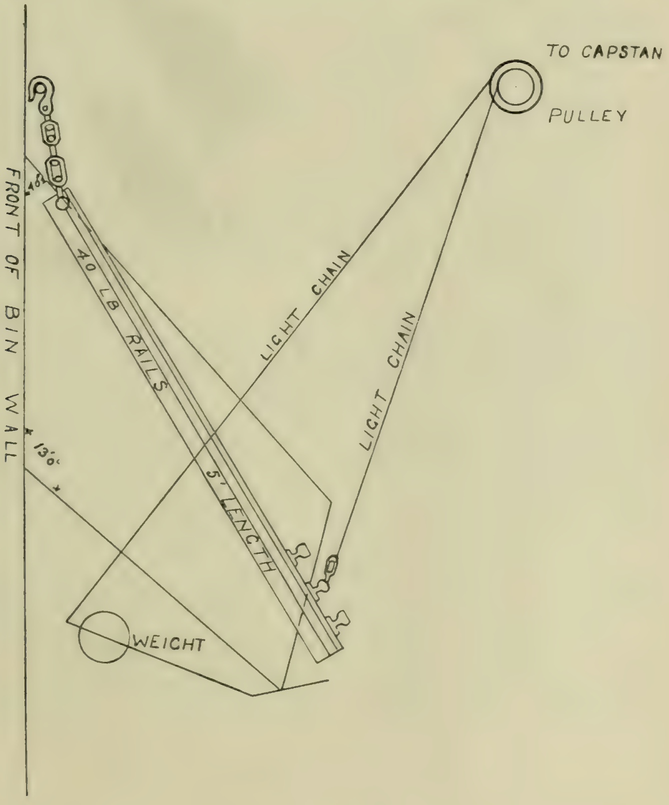


FIG. 10.
DISCHARGE APPLIANCE.

The floor of the bin is carried on twelve 10-in. x 10-in. oregon floor joists, running transversely to the caps of the foundation structure. The sides of the bin are carried on seven 10-in. x 10-in. oregon studs on the sides and end. The walls, both side and end, are strengthened by 6-in. stay bolts in each direction. As a further support, two 10-in. x 6-in. R. S. joists are placed longitudinally near the top of the bin. The sides and floor of the bins are made of 10-in. x 4-in. oregon, the bottom, or floor, being lined with $\frac{3}{8}$-in. plate and the sides with $\frac{1}{4}$-in. plate. The internal dimensions of the bin proper are 24 ft. 6 in. x 22 ft. 6 in. x 20 ft. deep. The floor of each bin builds up with material until it forms, by its angle of repose, a bottom which prevents undue wear on the bin.

In the front of each bin are four chutes so spaced as to discharge direct into the charge trucks, two chutes to each truck of the pair. The discharge of material from each chute is regulated by the appliance illustrated in Fig. 10.

Hanging in the chute proper is a grid composed of four lengths of 40-lb. railway iron, spaced at about 4 in., and held together by transverse lengths of rails at the front end and a bolt at top end. The grid is hung by a chain so as to have free movement. Attached to the front end is a light chain, which passes over a pulley and is attached to a capstan. The bottom of the chute has an extension hinged flap which, by means of a counterweight, moves on a hinge and blocks the front lip of the chute. To the lever is attached a light chain, which passes over a pulley and thence to the capstan. After passing over the first pulley the two chains are joined, so that only one goes to the capstan. The capstan is so fixed that the operator has full view of the charge weighbridge, and by manipulating the capstan can operate the railway-iron grid and movable lip in unison.

DELIVERY OF MATERIAL TO BINS.

As indicated in details of bin construction, the top of the bins is 58 ft. 8 in. above ground level. In order to deliver the material into the bins, five aerial hoists, which span each bin, are used. These hoists are of sufficient length to allow, in the case of coke, limestone, and ironstone, good storage capacity between the smelters and the wharf.

Details of Aerial Hoists.—Each hoist is of standard pattern, and the following are the essential details :—

Length of span	From 331 ft. to 472 ft.
Type of rope	Locked coil aerial.
Diameter of rope	2⅛ in.
Type of mast	Lattice.
Height of mast	90 ft.
Diam. of hauling rope	¾ in.
Weight of average loaded skip	3½ tons.
Net weight carried	2¼ tons.
Average trips per hour	25.
Weight of material moved per hour ..	56¼ tons.
Weight of material moved in 8 hours ..	450 tons.
Speed of lift	220 ft. per min.
Height of lift	80 ft.
Speed of traverse	660 ft. per min.

At present four of the aerial hoists are operated by steam-driven winding engines. The other has been electrified, and it is proposed to instal electric-driven hoists throughout.

The following details of the hoisting engines are given :—

Location of Hoist.	Diam. of Cylinders.	Length of Stroke.	Diam. of Drums.
D. & L. coke and slag ..	12 ft.	24 ft.	4 ft. 6 in.
Converter (H. & H.) ..	18 ,,	30 ,,	5 ,, 3 ,,
Flux hoist	(Electrically driven)		

Details of Aerial Hoist Skips.—The skips have approximately the following capacity when handling the undermentioned materials :—

Material.	Capacity.
Sinter	1¾ to 2 tons.
Returned slag	2 tons.
Limestone	2 tons.
Ironstone	2 tons.

Fig. 11 shows the essential features of the skips.

The skips are 6 ft. long x 4 ft. wide x 18½ in. deep, and are made of ¼-in. steel plate. There are three reinforcing pieces placed on the inside of the tub proper. One 4-in. x 3-in. x ½-in. T, placed 3 ft. centres from each end, and two flat 2-in. x ½-in. W.I. straps evenly spaced on both sides of the central T piece.

The yoke, which is permanently attached to the lifting gear of the aerial hoist, is made of a 7-in. x 2-in. piece of ironbark timber plated with W.I. The yoke is 6 ft. 10 in. over all, and from each end are hooks for connecting with the 2-in. diam. stud shown above. These hooks are of 1¾-in. diam. steel, and, where they attach to the skip, have a ¾-in. set on each side. Let into one of the attaching hooks of the yoke is a trigger, which drops into the recess on the end extension of the skips. This holds the skip while in operation, and the binman discharges the skip by releasing the trigger by means of a rope, when the skip being hung below its centre of gravity, overturns.

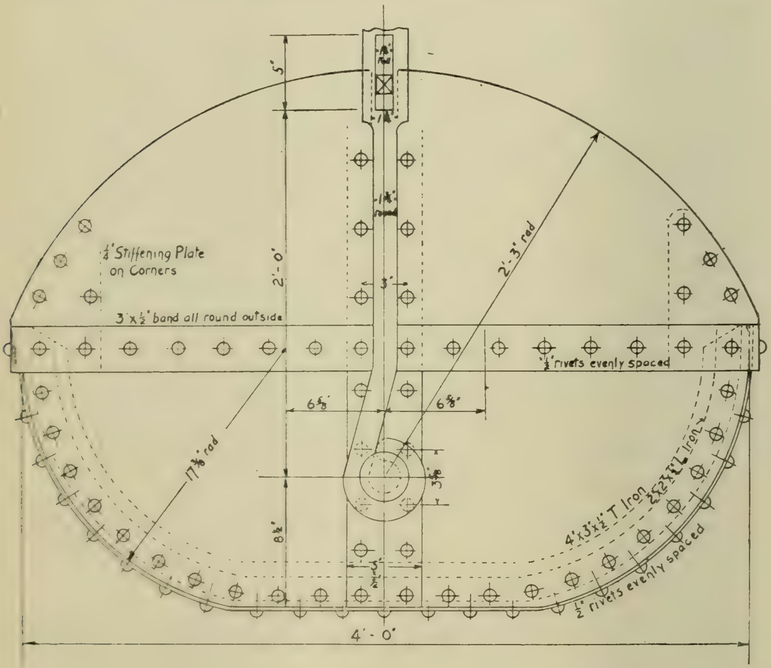


FIG. 11.
FEATURES OF SKIPS.

HANDLING OF COKE.

The major portion of the coke supply is obtained from the Company's own coke works, situated at Bellambi, in the South Coast District of New South Wales. It is freighted from Port Kembla to Port Pirie in vessels which average 3100 tons coke per cargo.

The following data applies to the unloading of coke from ships :—

Average number of hatches in ship	4 large and bunkers.
No. of gangs to large hatch	2.
No. of gangs to bunkers	1.
Strength of gang	6 shovellers, 2 plankmen, 1 winchman.
Tonnage discharged per gang	Average 10 tons per hour.

The coke is loaded into wicker-work baskets, five of which go to the ton. These baskets are delivered by ships' winches to a trolley running on portable trestle-work runway. They discharge direct to the coke tub of aerial hoist, and are then elevated to stack. The coke tubs are 8 ft. x 4 ft. x 4 ft. in dimension, and are mounted on a low-set trolley, on which they are delivered to the aerial hoist loading station. The available storage area for coke has a capacity of about 10,000 tons.

When a coke boat is unloading, the current coke supply for the smelters is delivered direct from ship to bin. The usual strength of labour gang handling coke from stock pile to smelter bin is as under :—

6 shovellers, 1 hooker-on, 1 binman.

The gang works from 8 a.m. to 5 p.m. daily, with an hour for lunch. On Saturday they cease work at noon. The storage capacity of the coke bin is just sufficient to hold over the period from 12 noon Saturday to 8 a.m. Monday.

HANDLING OF LIMESTONE AND IRONSTONE.

The limestone is obtained from either the Wardang Island or Point Turton, at which localities the Company has its own quarries. Ironstone is obtained from the B. H. P. Co.'s ironstone quarries situated at Iron Knob. These materials are lightered to Port Pirie in barges carrying 450 to 500 tons.

Only one hatch is worked at a time, the average gang being eight men in hold shovelling and four men on top. The material is filled into ore tubs holding about 25 cwt. These tubs are hung from below their centre of gravity, and, on release of a catch, dump their load into aerial hoist skips. These skips, containing about 2 tons 5 cwt., are delivered by means of a 2-ft. gauge tram-line to the hooking-on station of the flux hoist.

Ironstone and limestone are discharged from lighters at the rate of about 26 tons per hour. The storage capacity for limestone under the flux hoist is about 4000 tons.

HANDLING OF SILICIOUS ORE.

Silicious ores are forwarded by various outside mines, and are purchased on the ordinary tariff basis. Those silicious ores which contain not more than 5 % S are trommelled through a ¼-in. mesh, the material passing through goes to the D. & L. plant, and the oversize, up to about a 3-in. ring, to the smelters.

HANDLING OF REFINERY DROSSES.

These drosses are delivered by the refinery to an elevated dross tipple, where they are broken down to convenient size after being tipped from the pot. The drosses are then loaded into the skips of the flux hoist, and by this means are delivered to one compartment of the subdivided smelter bin.

QUARRYING AND HANDLING RETURNED SLAG.

Returned slag to the furnaces is quarried from a section of the slag dumped during the earlier smelter campaigns. The face of slag at present being quarried is from 20 to 25 ft. deep.

The following comprise the slag pit gang:—

1 shift boss.
8 drill men.
12 shovellers.
1 powder-monkey.
4 boys (horse-drivers).
1 hoist hooker-on, with boy assistant.

The average depth of hole drilled is about 19 ft., and set about 17 to 18 ft. back from the face of the quarry. The average charge of explosive is 15 packets of gelignite. Bickford's fuse and No. 8 detonators are used. The hours of labour are 8 to 5, with one hour for crib on 5 days, and 8 to noon on Saturday. The output of the quarry is 200 tons per day.

From the rill of the face the slag is shovelled into aerial hoist skips placed on a tramway trolley, and is drawn by horses to the hooking-on station of the aerial hoist.

SECTION F.

PREPARATION AND DELIVERY OF CHARGE TO FURNACES.

The five smelter storage bins are in an alignment parallel to the centre line of the blast furnaces. In front of the bins runs a 3 ft. 6 in. gauge line carrying the charge trucks. A similar line runs over the centre line of the furnaces, the centres of these two parallel tracks being 36 ft. On either end of these parallel tracks are automatic mechanical traversers, which deliver a pair of charge trucks from the bin line to the furnace line, and in turn deliver the empty trucks to the bin line.

OUTLINE OF MECHANICAL TRAVERSER.

Running centrally over the bin line and the furnace line are hauling wires $\frac{7}{8}$-in. diam., travelling at a speed of 60 ft. per min. Since both these tracks are elevated in relation to the furnace feed floor, the return cable of each line is able to travel under the track platform. In that of the furnace line the return cable passes through a special protecting plate when it crosses each furnace, so that it is not damaged by the falling charge.

Attached to these respective ropes are studs $1\frac{3}{4}$ in. diam. x 2 in. long, spaced at 15-ft. centres. These studs engage with a slot on each pair of charge trucks, the slot being fixed with springs acting as a shock absorber. Immediately in front of this slot, and forming part of the charge truck, is a roller so mounted that by means of a small lever it can be moved in a vertical travel. Consequently, when the operator wishes to hold a truck at a certain position it is only necessary for him to raise the roller, which then engages the hauling rope and removes it from the fixed slot of the truck. The rope then travels over the roller, and the truck remains stationary.

Both hauling ropes are operated by a common motor, which drives a countershaft, which, by means of a system of gears, operates the main driving sheave of each line. The traverser at each end of the lines is simply a large truck carried on a wheel base of 6-ft. centres, and running on a track 23-ft. 6-in. gauge. The four running wheels of the traverser are double-flanged (see Fig. 12).

Action of Automatic Operations.—

(1) The truck standing at point "A" on bin side ready for crossing.

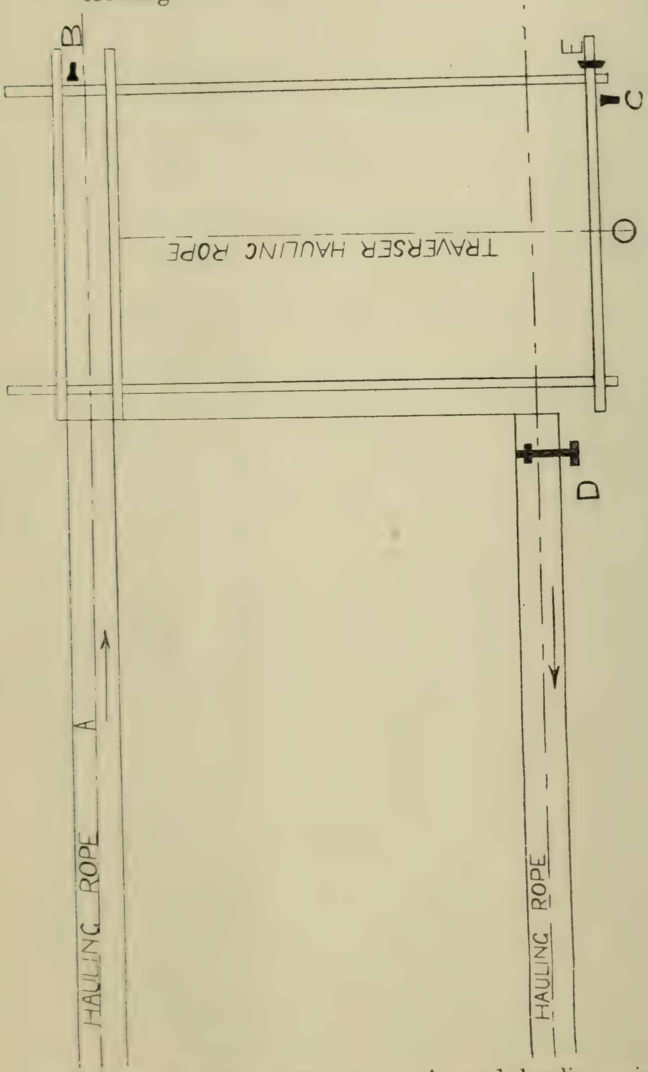


Fig. 12.
Plan of Traverser.

(2) Binman drops lever on truck, and hauling wire engages with truck.

(3) Truck hauled on to traverser.

(4) Truck hits buffer " B," which disengages hauling wire from the truck and operates the traversing lever forward.

(5) Traverser with truck crosses to furnace side.

(6) Traverser operates buffer " C," which drops hauling wire into position.

(7) Trucks are hauled from traverser.

(8) Trucks operate lever " D," which in turn operates valve at point " E " to reverse.

(9) Traverser returns to starting-point on bin side.

(10) The action of traverser leaving furnace side releases buffer " C," and raises hauling wire on furnace side.

The hydraulic service is the power which operates the installation.

The average time taken for the traverser to cross and return is about 2 min.

CHARGE TRUCKS.

There are 26 trucks on the charge floor, the trucks being coupled in pairs by means of 5-in. x 4-in. T iron.

The following are the chief dimensions of the charge trucks :—

Length over all, 13 ft. ; width over all, 5 ft. 3 in. ; height over all, 3 ft. 2 in.

Wheel base, 9 ft. 10 in.

Internal measurements.—Top, 9 ft. long x 5 ft. wide.
Bottom, 8 ft. long x 2½ ft. wide.
Depth, 2 ft. 8 in.

The body of the truck is made of ¼-in. steel plate. The trucks are hopper-bottomed type, the discharge being governed by four doors operated by one lever. These four doors are hinged on the longitudinal centre line of the truck, thus giving an outward discharge on both sides. The hinges for these doors are carried on a centre plate or T piece placed vertically and supported in transverse section by three plates of similar section on each side. This system of reinforcing also has the dual service of dividing the bottom of the truck into eight areas—four areas per pair of doors.

Since it often happens that the charge in the truck needs to be dumped in a special part of the furnace shaft, the doors can be operated in pairs by the furnace man holding a block under one pair. The truck is then moved forward or back, as the case may be, and the other section of the truck discharged. The division of the truck bottom into eight sections as a grid assists this operation.

CHARGE WEIGHBRIDGES.

To each smelter bin are two Pooley and Sons' weighbridges of multiple beam type. They have a maximum capacity of 8 tons, and a minimum capacity of 7 lb. They are at such centres that when a pair of charge trucks are at a bin one truck of the pair is on one weighbridge.

PREPARATION OF CHARGE.

There are four chutes in front of each bin through which the material is delivered to a pair of charge trucks standing on weighbridges. Two chutes deliver into each truck, giving a fairly even distribution and obviating much labour in levelling. There is one bin attendant to each bin, and by means of the chute control described under "Bins" ("Delivery of Ores, etc., to Furnaces"), the attendant can operate each chute while keeping in front of the weighbridge. When the requsiite weight of material is shown on the weighbridge the bin man closes the chutes, levels the charge, if necessary, and then engages the cars to the hauling cable, which conveys them to the next bin.

In this way the trucks pass from bin to bin until the charge is completed, and finally under a fresh-water spray to prevent dusting of charge and to keep the top of the furnace cool.

The sequence of the bins in making up the charge is—

(1) H. & H. sinter bin.
(2) D. & L. sinter bin.
(3) Refinery dross compartment (limestone, silicious ore, ironstone, as case may be).
(4) Returned pit slag.
(5) Coke.

In case a furnace requires feeding on one side more than the other, an apron fixed to one side of the truck by hooks is placed in. This apron is sufficiently long to lie along the central longitudinal partition, and consequently gives a charge which can only discharge through one pair of side doors.

DELIVERY OF CHARGE TO FURNACE.

The central flue, which is 3 ft. wide, extends throughout the length of the furnace. This flue, having a gabled roof protected by baffle plates ⅜-in. thick, acts as a distributor for the charge in addition to its use as a flue. The furnaces having open tops, the charge, falling from the baffle plates on the gabled roof of the central flue, is directed into the furnace by a fender on each side, extending throughout the length of the furnace. They are in two lengths, and are readily portable, so as to allow freedom when a furnace is being barred.

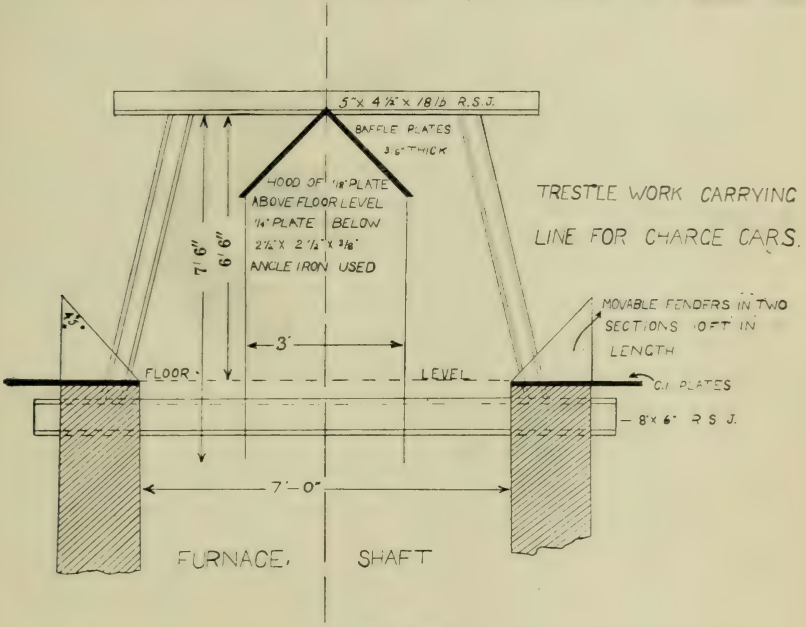


FIG. 13.

SECTION G.

BLAST FURNACE CHARGE DATA.

The blast furnace operations depend directly upon the sinter produced in either the D. & L. or H. & H. sections of the blast-roasting department fulfilling the following esssential conditions :—

(*a*) Having a low sulphur content.

(*b*) Being strong as regards physical condition.

Experience has shown that in order to attain these conditions it is essential that the blast roasting department has freedom of control as regards the composition of the charge to be sintered. Normally, it is found that the sintered material is well balanced as regards fluxing constituents, and it is interesting to note that the blast furnaces have at various times operated over lengthy periods upon sintered material alone (excluding the addition of returned slag). However, at periods it is necessary for the SiO_2, FeO, or CaO content of the sintered material to be adjusted, either individually or collectively. This is accomplished by the addition on the feed floor of silicious ore, coarse ironstone, and limestone.

For the past twelve-monthly period the blast-roasting department has deleted ironstone from the charge, thereby effecting the saving in the cost of crushing this constituent.

In order to indicate the quality of the sintered material, five typical blast roasting charges are given, together with the analysis of the D. & L. and H. & H. sinters produced from these charges :—

Typical Blast Roasting Charges.

Charge No.	% Coarse Concentrate.	% Slime Concentrate.	% Lime Sand.	% Silicious Ore.	% "Risdon" Residue.	% Granulated Slag.
1	42	29	5	2	12	10
2	36	36	6	2	10	10
3	24	51	4	2	9	10
4	21	49	7	6	5	12
5	42	22	8	7	9	12

The major slag-forming constituents typical of the above individual components of the blast roasting charge are indicated in the following table :—

BLAST ROASTING CHARGE.

Slag-forming Constituents of Charge Components.

Component.	% SiO_2.	% FeO	% MnO.	% CaO.	% MgO.	% Zn.
Coarse concentrates	5.6	3.4	2.37	0.60	0.50	7.3
Slime concentrates	7.1	4.1	1.4	1.10	0.30	8.5
Lime sand ..	6.1	—	—	47.0	3.5	—
Silicious ore ..	43.6	7.7	5.7	0.9	0.2	3.0
"Risdon" residue	8.1	22.43	4.10	0.93	1.11	—
Granulated slag ..	20.7	28.0	4.5	11.6	2.0	17.0

The component "Risdon" residue is the residue obtained from the Electrolytic Zinc Co. of Australia, and contains lead and silver values which are not extracted in their process.

Sinter from Charge No.	DWIGHT & LLOYD.							HUNTINGTON—HEBERLEIN.						
	% Pb.	% SiO_2.	% FeO.	% MnO.	% CaO.	% Zn.	% S.	% Pb.	% SiO_2.	% FeO.	% MnO.	% CaO.	% Zn.	% S.
1	44.0	8.8	10.4	2.5	6.2	12.6	2.4	43.5	9.8	9.7	2.4	5.9	11.0	2.6
2	43.8	8.1	8.2	2.7	6.8	11.7	2.4	43.4	9.0	9.0	2.9	6.6	11.3	2.5
3	42.4	9.4	10.0	2.4	5.8	13.5	2.2	42.5	9.6	9.5	2.5	5.6	13.1	2.8
4	43.7	8.4	9.2	3.0	6.9	11.6	2.3	43.2	8.9	9.0	2.7	6.8	11.2	2.1
5	44.1	8.6	10.1	3.4	6.9	10.5	2.1	44.5	8.6	10.0	3.4	6.8	9.8	2.2

The analysis of the above sinters is the average of a weeks' production when the above typical charges were being treated in the blast-roasting department.

In order to explain the similarity in analysis of the D. & L. and H. & H. sinter, it is necessary to note that one set of D. & L. machines as a pre-roasting unit is used.

The pre-roasted material is then finally roasted in—

(*a*) A second unit of the D. & L. machines.

(*b*) The H. & H. pots, which were in commission prior to the installation of the D. & L. machines.

The pre-roast section has a greater capacity in pre-roasted material than can be handled on the final roasting D. & L. machines, and this surplus is treated in the H. & H. pots.

Each type of sinter is practically identical as regards chemical composition, but both D. & L. and H. & H. sinter possess the following advantages:—

(*a*) The D. & L. sinter costs less to produce in the ratio of as 1 is to 1.3.

(*b*) The H. & H. sinter is physically of a much stronger nature, and keeps an open charge, with consequent increase in furnace speed.

The following blast furnace charge may be taken as typical of normal operations :—

	lb.		%
D. & L. sinter	2600,	equal to	65
H. & H. sinter	1300,	,, ,,	32.5
Limestone or ironstone ..	100,	,, ,,	2.5
Total charge	4000,	equal to	100
Returned slag	900,	,, ,,	22.5
Coke	480,	,, ,,	12.0

As regards the ratio of D. & L. to H. & H. sinter on the charge, it is to be noted that this ratio is adjusted daily according to the supply of each class of sinter in the respective smelter bins.

Considering the charge on the basis of material treated in a half-yearly period, the following data is of interest :—

Ratio D. & L. sinter treated to H. & H. sinter—2 : 1.

—	% on Basis of Charge.	% on Ton Input Lead.
% Limestone flux added	6.69	16.41
% Ironstone flux added	1.97	4.84
% Silicious ore added	4.09	10.03
% Returned slag added	39.97	97.97
% Coke	13.51	33.12
Tons total slag per ton input lead ..	1.73	
Tons new slag per ton input lead ..	1.06	

Reviewing the above data, it must be noted that, as regards limestone and silicious ore on the charge, the above figures comprise limestone and silicious ore added in blast roasting charge plus that added on blast furnace feed floor.

Returned slag comprises granulated slag added in blast roasting charge plus returned slag added on blast furnace feed floor, plus returned slag added while the furnaces are being barred (see notes on Procedure in Barring a Furnace).

The figures given relating to coke comprise the normal percentage of coke added in charge plus extra coke added while a furnace is barring.

Regarding ironstone, the above percentage refers solely to this material added on the blast furnace feed floor, since, as previously indicated, this material has been deleted from the blast roasting charge.

RETURNED SLAG ON CHARGE.

The essential reasons for adding returned slag on the blast furnace feed floor are :—

(1) To increase the slag flow, thereby ensuring better conditions at the tap-hole.

(2) Being the most infusible constituent on the charge, it tends to keep the fire down, thereby counteracting the tendency for furnace to have "hot top."

(3) To keep the charge open.

As regards the recovery of lead values from returned slag, it is considered problematic whether the use of returned slag in this aspect has any economic value. The lead in returned slag is largely present as lead sulphide, and there are, as well, metallic particles, which latter are recoverable.

The returned slag used varies slightly in composition as compared with the slag at present produced. There is but slight and unimportant variation in the SiO_2, FeO, CaO content, the only marked difference being in the ZnO content, that in the returned slag being around 14 %, and that in current slag around 21 %.

In the writers' opinion, under the reducing conditions existing in the lead blast furnace, the sulphide of lead contained in the returned slag is not reduced in any considerable extent to metallic lead in its passage through the furnace.

As bearing on this point, the following results are of interest :—

Case No. 1.—With three furnaces operating, a test was carried out to determine the influence of returned slag on the lead content of the final or total slag leaving the furnaces. The general operating conditions on all furnaces were as identical as possible in practice.

The charge to the furnaces was :—

—	No. 1 Furnace.	Nos. 2 and 3 Furnaces.
	lb.	lb.
D. & L. sinter	2700	2700
H. & H. sinter	1200	1200
Ironstone	100	100
Returned slag	nil	500
Coke	500	500

The average lead content of the slag from each furnace over a 24-hour period was :—

	%
No. 1 furnace	1.2
No. 2 furnace	2.6
No. 3 furnace	2.8

The returned slag added to Nos. 2 and 3 furnaces contained 4.3 % Pb.

Figured on the basis of original slag from No. 1 furnace to original slag plus added slag on Nos. 2 and 3 furnaces, it was shown that the variation in Pb in slag from Nos. 2 and 3 furnaces over that from No. 1 furnace was accounted for by the added slag.

It is interesting to note that at the end of this test period No. 1 furnace showed a marked increase over Nos. 2 and 3 furnaces as regards "hot top."

Case No. 2.—On another occasion, with three furnaces operating under good and practically identical conditions, the slag from each furnace showed the following abnormally high percentage of lead :—

	%
No. 1 furnace	3.7
No. 2 furnace	4.2
No. 3 furnace	3.9

Now, the returned slag added to the charge on this date contained an abnormal percentage of lead—*i.e.* ,9.5 %. This indicated the cause of the high lead content of the furnace slags. The ratio of new slag to added slag on the charge was 2 parts new slag to 0.7 parts added slag.

Take the new slag as containing a normal percentage of Pb—*i.e.*, 1.8 %. We then have 2 parts new slag at 1.8 % Pb, and 0.7 parts added slag at 9.5 % Pb, giving 2.7 parts total slag containing 10.1 % Pb, or one part total slag containing 3.7 %, which figure agrees closely with the actual loss in the slag.

ANALYSIS OF FLUXES, COKE, ETC.

The following are the average analyses made over a half-yearly period on the various fluxes and materials forming part of the regular blast furnace charge :—

Analysis of Coke.	%
H_2O	1.07
V. H. C.	1.67
F. C.	79.26
Ash	18.0
S	0.66

Analysis of Coke Ash.	%
SiO_2	45.5
Fe_2O_3	10.4
MnO	1.0
CaO	5.9
MgO	1.8
Al_2O_3	33.6
SO_3	0.75

Analysis of Limestone.—

	Wardang Island.	Point Turton.
	%	%
SiO_2	6.8	2.70
Fe_2O_3	2.29	0.75
Al_2O_3	2.00	0.41
CaO	42.38	52.00
MgO	5.72	1.08
Cl	0.40	0.03
SO_3	0.25	0.115
P_2O_5	0.28	0.155
H_2O	0.75	0.30
Organic	0.39	0.20

Analysis of Ironstone.	%
SiO_2	2.61
Fe_2O_3	91.31
FeO	82.18
Fe	63.92
Mn_3O_4	0.85
Mn	0.61
CaO	0.37
MgO	0.65
Al_2O_3	1.82
H_2O	1.28

Analysis of Baghouse Material (Sintered).	%
Pb	67.4
Pb as $PbCl_2$	4.0
Ag	1.4 oz.
Cu	0.083
Zn	7.9
S	7.17
S as SO_4	2.77
Sb	0.25
As	0.48
Cd	2.12
Cl	2.32

SECTION H.

SMELTER BULLION.

The following typical analyses show the composition of normal smelter bullion over weekly periods :—

	No. 1.	No. 2.	No. 3.	No. 4.
Ag	65.0 oz.	84.0 oz.	83.5 oz.	66.7 oz.
Cu	0.92 %	0.78 %	0.58 %	1.04 %
As	0.41 %	0.38 %	0.63 %	0.21 %
Sb	0.65 %	0.73 %	0.80 %	0.90 %

The variation in the above constituents of the smelter bullion is influenced largely by the ratio of granular to slimes concentrates treated in the blast roasting charge. The average slime concentrates carries higher silver values than is the case with granular concentrates, and, again, the percentage of impurities such as Cu, As, and Sb is higher in the slime concentrates.

The smelter bullion carries out with it from the furnaces, in a dissolved state, two classes of dross, which are given off on cooling, forming a thin crust or scum on the surface of the bars. In the refining process these drosses are eliminated in two stages preparatory to the desilverizing of the bullion, and we obtain two products—

(1) A black dross, which is given off in the copper-softening furnace, and which amounts to about 8 % of the original bullion.

(2) A metallic dross, or metallic alloy, which is obtained from the desilverizing kettles preparatory to the zincing operation. This dross is about 6 % of the softened bullion.

The following analyses are typical of the drosses obtained from the smelter bullion in the above stages of the refining process :—

	Dross ex Copper Softeners.	Dross ex Zinc Pans.
	%	%
Pb	72.2	93.0
Cu	10.83	3.79
As	2.47	0.86
Sb	0.83	1.27
Fe	0.90	0.007
Zn	2.81	0.30
S	5.10	nil
Ag	30.0 oz.	68.4 oz.

On the blast furnaces the existing practice is to mould the smelter bullion into bars, which are then transported to the refinery.

However, it is interesting to point out that a complete installation has been designed by which the smelter bullion will be transported to the refinery in a molten condition in ladles of 2 to 3 tons capacity. These ladles will be transported to the refinery by a telpher system.

BULLION MOULDS.

Running parallel to the length of the furnace is an 18-in. gauge track, the inner rail of which is 2 ft. from the side of the crucible tank. The track is cast solid in cast-iron, three lengths to the side of the furnace. On the outer edge of this casting is a rack with teeth 4 in. long x 1 in. sq. On this 18-in. gauge line run two bullion mould trucks, one to each lead well. The trucks are tabletop trucks, on each of which is fixed 20 bullion moulds. They are held together longitudinally by $\frac{7}{8}$-in. strapping bolts, and to the truck body by hooks on each side. By this arrangement the bullion moulds are solid and compact, and yet a broken mould can be readily replaced.

The rack on the outside of the other rail is for pushing the truck along when bullion is being moulded. The tops of the bullion moulds, when mounted on the track, are 1 ft. 10 in. above rail level.

Before bullion is moulded the moulds are swabbed out with crude petroleum oil or dusted with fine coal-dust, in order to prevent the bullion sticking to the mould.

At present two types of bullion moulds are in use—

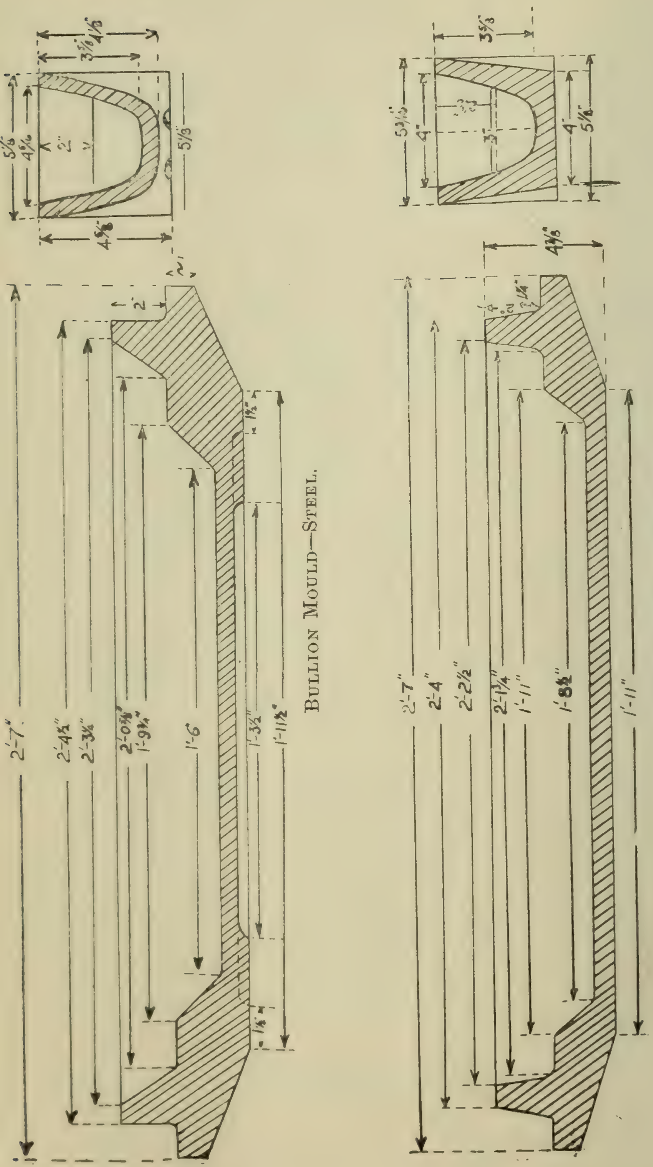
(1) Cast iron. (2) Cast steel.

Taken over an average half-yearly period, with 182 moulds in constant use (all moulds of cast-iron), the average life per mould was 29 days.

Comparing cast-iron and cast-steel bullion moulds, the following figures are of interest:—

(1) Cost of steel moulds compared with cast-iron moulds is as 2.2 is to 1.

(2) Life of steel moulds compared with cast-iron moulds is as 4 is to 1.



Bullion Mould—Steel.

Bullion Mould—Cast-iron.

Fig. 14.

SECTION I.

BLAST FURNACE SLAG.

In dealing with the subject-matter of blast furnace slags it is the intention in this article to adopt a practical rather than a theoretical viewpoint.

The slag typical of the Broken Hill Associated Smelters Pty. Ltd. practice varies from the normal lead blast furnace slag markedly in its high ZnO content. Standard authorities claim that 10 % to 12 % is the economic limit of ZnO which can be carried in the slag.

From the practical viewpoint it is essential that the blast furnace slag shall comply with the following conditions :—

(1) Liquidity.
(2) Low specific gravity.
(3) Fusibility within economic practice.

It has been clearly demonstrated at the Port Pirie works that these essentials can be economically maintained in the production of a slag averaging over half-yearly periods a ZnO content of around 22 %, and over shorter periods an average of from 25.5 % to 33.3 % ZnO.

In dealing with the ZnO constituent of the blast furnace slag, it is interesting to follow the development of our general practice.

During the years 1917 and 1918 the blast roasting practice had not reached its present high standard, and the sulphur content of the sintered material treated in the blast furnaces was much higher than that being at present treated.

While treating this early sintered material, considerable trouble was experienced in the blast furnace operations when the ZnO in the slag exceeded 15 %. The following difficulties were experienced—

(1) The tap-holes were very hard to open.
(2) The tuyeres were dull and hard.
(3) The slag generally appeared short, and possessed a very high fusion point.
(4) A large amount of "surface matte" was produced, which floated on top of the slag by reason of its cellular structure.
(5) A smaller quantity of "bottom matte" was produced.

As regards the composition of "surface" and "bottom" matte, analysis of typical samples showed very little difference. This is indicated in the following analyses:—

——	Surface Matte	Bottom Matte.
	%	%
SiO_2	3.1	3.4
FeO	45.4	48.53
MnO	0.9	4.0
CaO	1.8	1.8
MgO	0.56	0.28
Al_2O_3	0.25	0.94
Zn	8.6	7.90
S	15.2	15.20
Cu	0.17	1.07
As	2.33	0.37
Sb	0.15	0.04
Pb..	7.8	4.7
Ag..	11.0 oz.	3.2 oz.

Recent development in the blast-roasting practice has resulted in the production of a sinter carrying a much lower sulphur content than could be previously maintained. Consequently, at the present time B. H. A. S. normal blast furnace slag carries 22 % ZnO, and much higher percentages (up to 33 % ZnO) have been carried successfully without serious furnace troubles.

In all cases the slags were perfectly fluid; they gave no trouble at the tap-hole; the tuyeres remained good; a good separation took place in the crucible; and no surface matte was made. In order to achieve these objects, the following conditions were maintained:—

(1) Low input sulphur in the charge (essential).
(2) Absence of sulphide of Zn in the sintered material.
(3) High FeO + MnO.
(4) Low CaO + MgO.

Provided no ZnS is present in the sintered material, the sulphur content of the charge may be considerably higher, and yet no surface matte be made.

From the foregoing remarks it is evident that the solution of the problem of carrying high Zn in blast-furnace slags is directly dependent on the efficiency of the blast-roasting practice. The experience gained at the Port Pirie works clearly indicates that with low sulphur in the sintered material high percentage of zinc can be carried in the slag without difficulty. It can be taken definitely that, from a practical standpoint, zinc (unless present as zinc sulphide) has no detrimental effect as regards blast furnace slags within the limits of ZnO content shown in Port Pirie practice.

The type of slag aimed at is one which is low in SiO_2 and CaO and high in FeO, but in which the CaO, MgO + FeO, MnO is more than sufficient to satisfy all the SiO_2 on a mono-silicate basis, leaving excess iron as a carrier of ZnO. In practice, however, it is not always economical to produce slags of a given type, but the following typical analyses indicate the ranges which have been successfully run at Port Pirie over such periods of time as to eliminate possible interference with economic practice :—

Analysis of High ZnO Slags :

——	No. 1.	No. 2.	No. 3.	No. 4.
	%	%	%	%
SiO_2	18.4	19.5	22.5	20.8
FeO	20.3	26.2	30.2	33.5
MnO	5.1	4.8	4.8	5.1
CaO	8.9	9.9	8.7	7.8
ZnO	33.3	25.5	18.4	15.7

The sulphur content of these slags is usually 3 %, and it is probable that small amounts of zinc exist as zinc sulphide.

Analysis of Slag over Half-yearly Period.—The following analysis represents the average of a daily aliquot part sample over a half-yearly period :—

	%
SiO_2	20.7
FeO	28.0
MnO	4.5
CaO	11.6
MgO	2.0
Al_2O_3	6.2
ZnO	20.92 = 16.8 % Zn
S	2.7
Cu	0.19
As	0.02
Sb	0.05
Cl	0.077

The following data is typical of Port Pirie blast-furnace slags :—

(1) Specific gravity, 3.66.

(2) Specific heat, 0.1767.

(3) Temperature at tap-hole, 1000° C. to 1200° C.

SECTION J.

SLAG GRANULATION—GENERAL LAY-OUT.

The five 120-ton furnaces are each equipped with a separate granulating installation. Fig. 16 shows the general lay-out at each furnace up to the point of delivery of the slag to the boot elevator, from which it is delivered to a 150-ton storage bin.

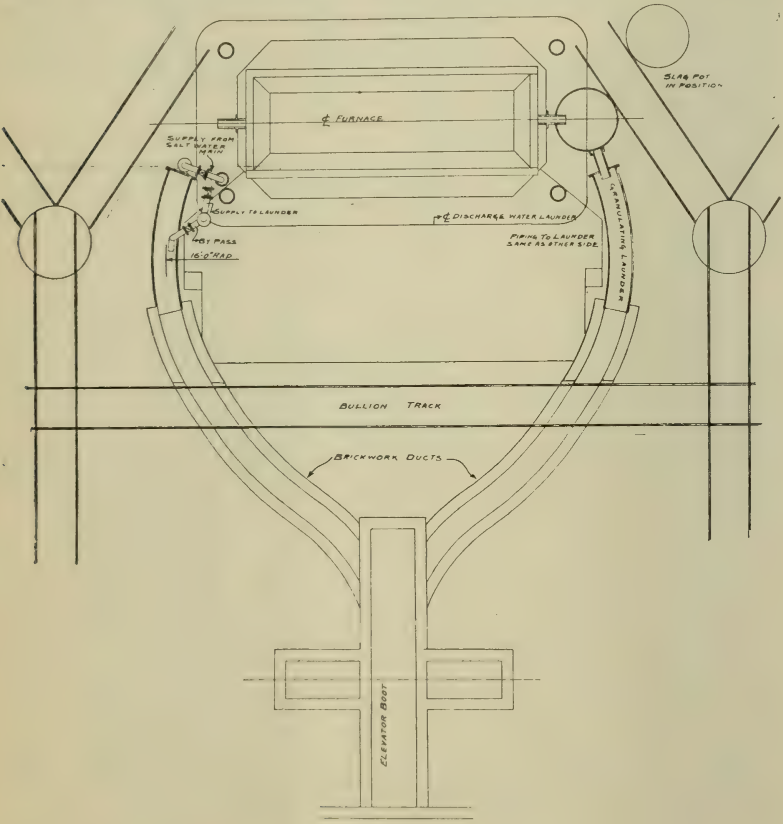


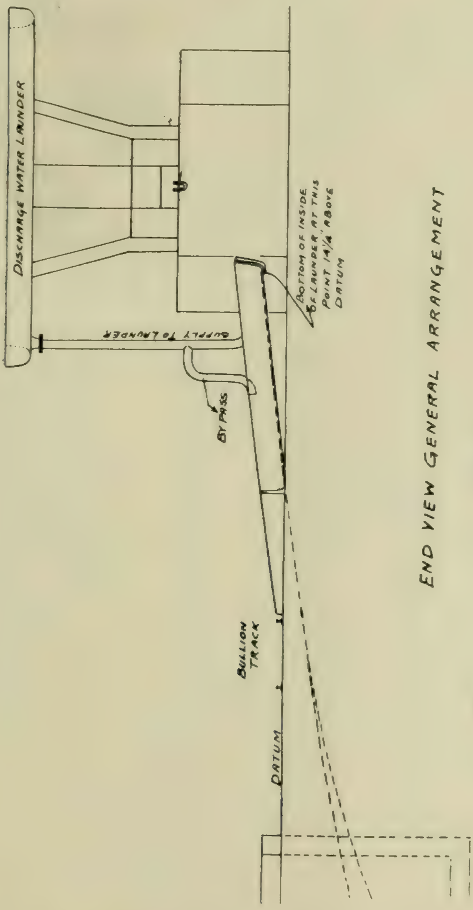
FIG. 15.
PLAN—GENERAL ARRANGEMENT.

The slag, tapped from either end of the furnace, first flows into a settler, and, overflowing from this, the stream of slag falls a vertical height of 18 in. to the bottom of the granulating launder. Before reaching the bottom of the granulating launder a jet of water impinges on the falling stream of slag, when the latter is quickly granulated.

The water for granulating is obtained from two sources :—

(1) Overflow jacket water.

(2) Auxiliary water obtained from salt-water supply tanks at a pressure of 35 ft. mean head.



END VIEW GENERAL ARRANGEMENT

FIG. 16.

The supply of the latter water is controlled by a direct-action valve, so that it is only used when granulating is taking place.

During the granulating process dense volumes of steam are generated. To cope with this, the granulating launder is covered in, and the steam, etc., led off by one stack midway in the length, and a larger stack built over the boot of the elevator just where the two launders discharge.

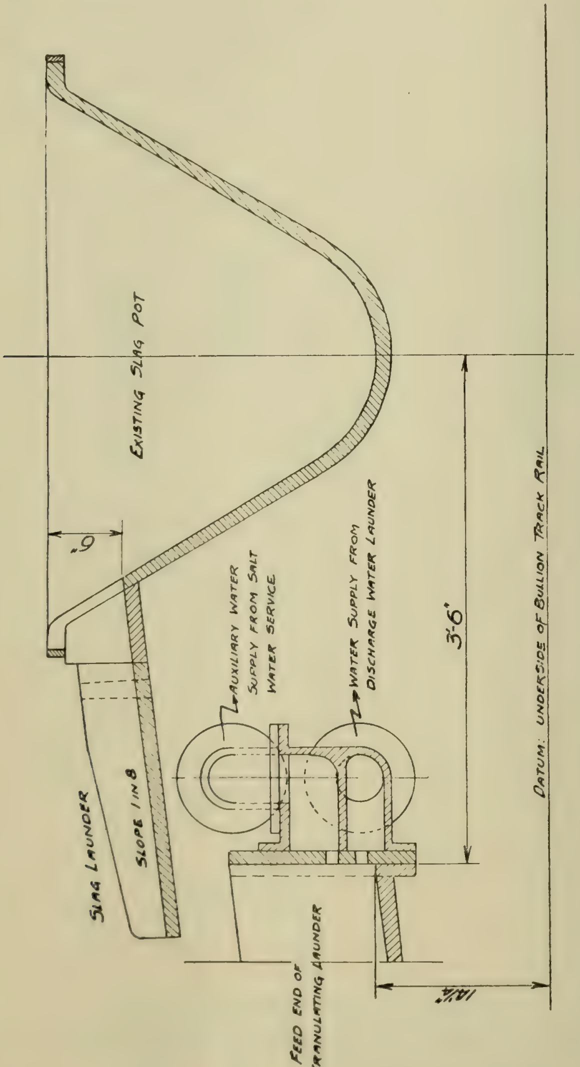

EXISTING SLAG POT
6"
3'-6"
DATUM: UNDERSIDE OF BULLION TRACK RAIL
WATER SUPPLY FROM DISCHARGE WATER LAUNDER
AUXILIARY WATER SUPPLY FROM SALT WATER SERVICE
SLOPE 1 IN 8
SLAG LAUNDER
FEED END OF GRANULATING LAUNDER
14¼"

FIG. 17.

SHOWING RELATION BETWEEN SLAG LAUNDER AND GRANULATING LAUNDER.

The slag is then elevated to a 150-ton storage bin, which is built to such a height as to allow the 3-ft. 6-in. gauge locomotive to pass

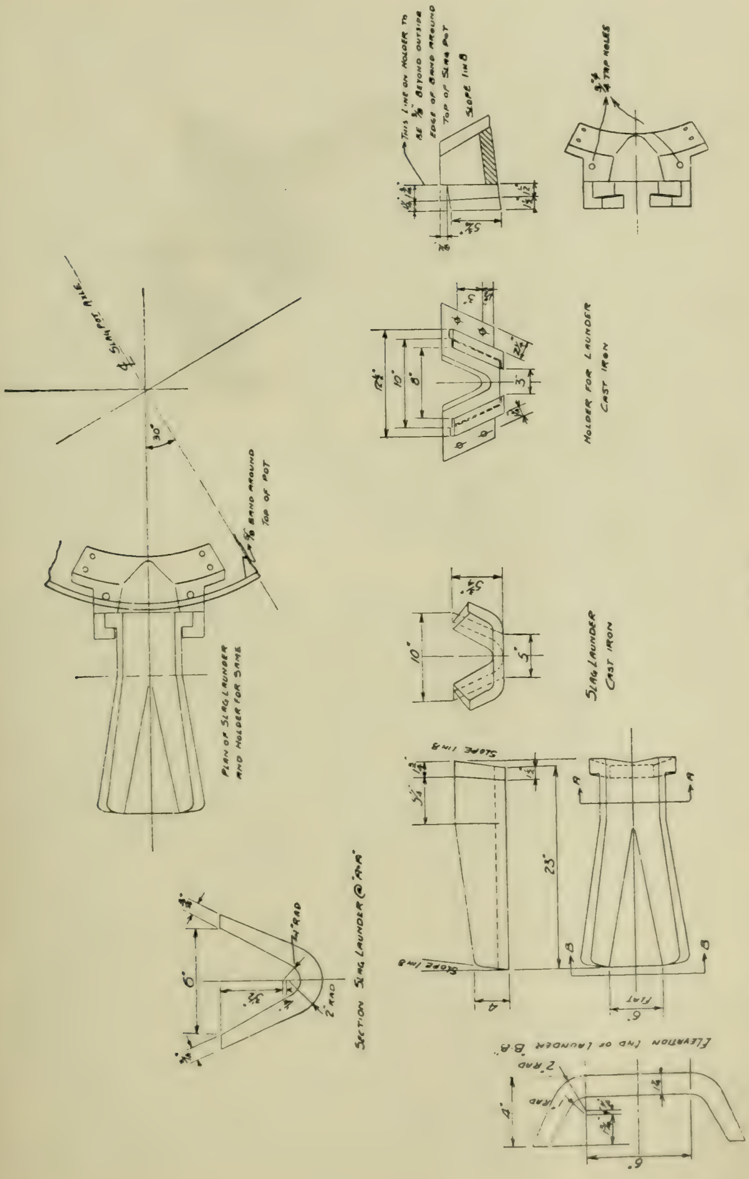

SLAG POT AXLE
30°
3/8" BAND AROUND TOP OF POT
PLAN OF SLAG LAUNDER AND HOLDER FOR SAME
SECTION SLAG LAUNDER @ "A-A"
6"
3/4" RAD
2" RAD
SLAG LAUNDER CAST IRON
10"
5"
HOLDER FOR LAUNDER CAST IRON
12¼"
10"
8"
3"
THIS LINE ON HOLDER TO BE 3/8" BEYOND OUTSIDE EDGE OF BAND AROUND TOP OF SLAG POT
SLOPE 1 IN 8
3/4" TAP HOLES
23"
FLAT
9"
4"
ELEVATION END OF LAUNDER "B-B"
2" RAD
1" RAD

FIG. 18.

DETAILS OF CONSTRUCTION OF SLAG LAUNDER.

under the discharge door. The overflow water from the elevator boot flows into a drain, common to the five furnaces, thence to a settling dam, and returns to the harbour.

DETAILS OF GRANULATING ACCESSORIES.

(*a*) *Settlers.*—For this purpose the existing horse-drawn slag pots were adapted. Each bowl has a capacity of 0.75 tons of slag, two bowls being mounted on the one carriage. Each bowl is so fitted that it is capable of having attached to it a slag launder in either of two positions. This arrangement ensures the slag launder being in correct relation to the granulating launder should the slag bowls be turned. Fig. 17 indicates the relation between the slag launder and the granulating launder.

(*b*) *Slag Launder and Fittings.*—The slag launder is so cast that, when placed in its seating attached to the settler, it has a slope of 1 in 8. The height between the lip of the slag launder and the bottom of the granulating launder is 18 in. Fig. 18 sets out clearly the details of construction.

(*c*) *Granulating Launder.*—The length of each granulating launder, from the intake to the discharge end is 31 ft. The grade is 1 in 8. The launders are encased in brick throughout. In order to cope with the curve, the launder is set with a good super-elevation. Since, in the granulating process, large volumes of steam are generated, it is necessary to control this factor. The sides of the granulating launder are extended vertically by 3/16 in. M.S. plate and covered with hinged doors, joining in mid-section of the launder and hinged to open outward. This arrangement makes a good flue to cope with the steam, etc., and yet allows ready inspection of launders and removal of any blockage.

The hood over each launder connects with a ventilating stack 60 ft. high and 15 in. in diam., which carries off a large volume of the generated steam, etc. The usual hood over the slag spout of the furnace has an extension which copes with the fume at the actual granulating end of the launder, since 3 ft. of this is not covered, as mentioned above.

(*d*) *Admission of Granulating Water.*—The intake end of the granulating launder is closed with a cast-iron end plate, details of which are given in Fig. 20. In this casting there are two water-admission openings, 1 in. wide x 6 in. long, on a curve of 3½-in. radius. On the back of this end plate is attached the water-breast, full details of which are shown in Fig. 20. This water-breast can cope with two independent water supplies, which are eventually delivered to the granulating launder through two 1-in. x 6 in. curved slots in the end plate. The water delivered to the granulating launder through the upper slot in the jacket overflow water. This has a head of about 10 ft., and flows continuously at the rate of approximately 125 gal. per min. The lower stream is delivered by a 4-in. pipe from the salt-water service main. This water has a mean head of about 35 ft., and is delivered at the rate of 200 gal. per min.

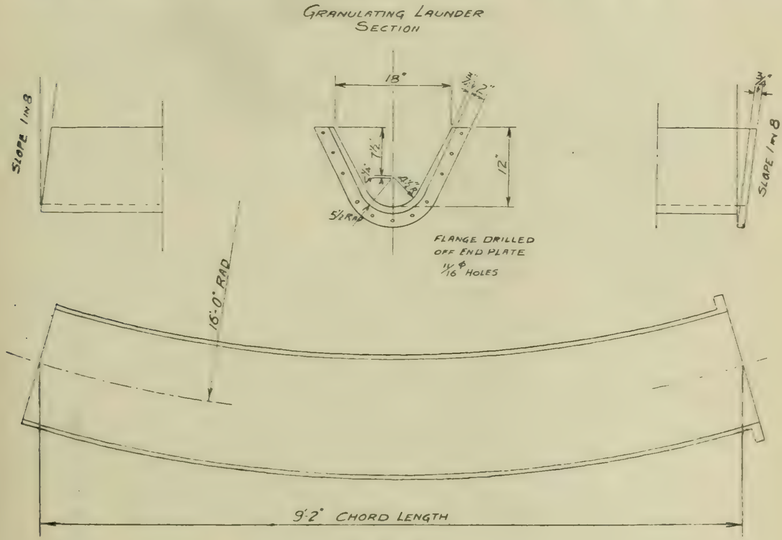


FIG. 19.
GRANULATING LAUNDER—CAST-IRON.

In order to control the use of this water a direct-action valve is installed, so that the water is only running when actual granulation is in process.

(*e*) *Elevator and Storage Bin.*—The elevators have a continuous rating of 24 tons per hour with buckets three-quarters full.

The elevator centres are at 52½ ft., the tail pulley being 42 in. diam. and 24 in. face, and the head pulley 36 in. diam. x 22½ in. face. The

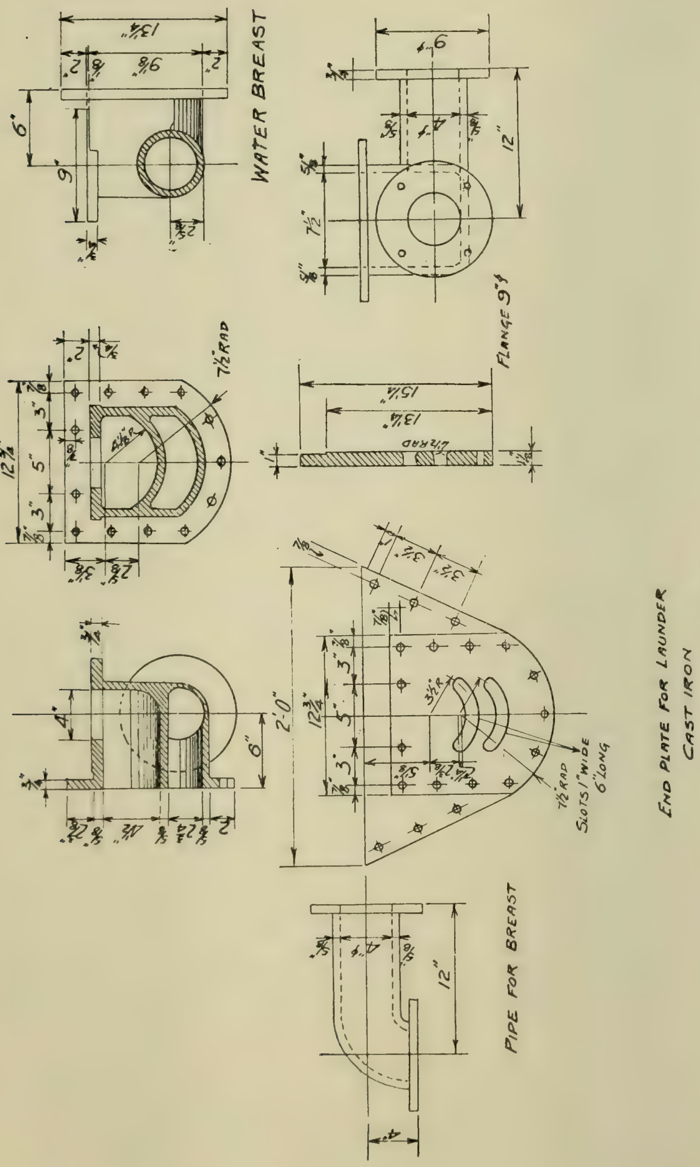

WATER BREAST
13¼"
9⅛"
2"
6"
9"
FLANGE 9"
12"
7½"
4"
15¼"
13¼"
7½" RAD
12¾"
5"
3"
4⅛" R
2'-0"
6"
3½" R
7½" RAD
SLOTS 1" WIDE 6" LONG
END PLATE FOR LAUNDER
CAST IRON
PIPE FOR BREAST

FIG. 20.

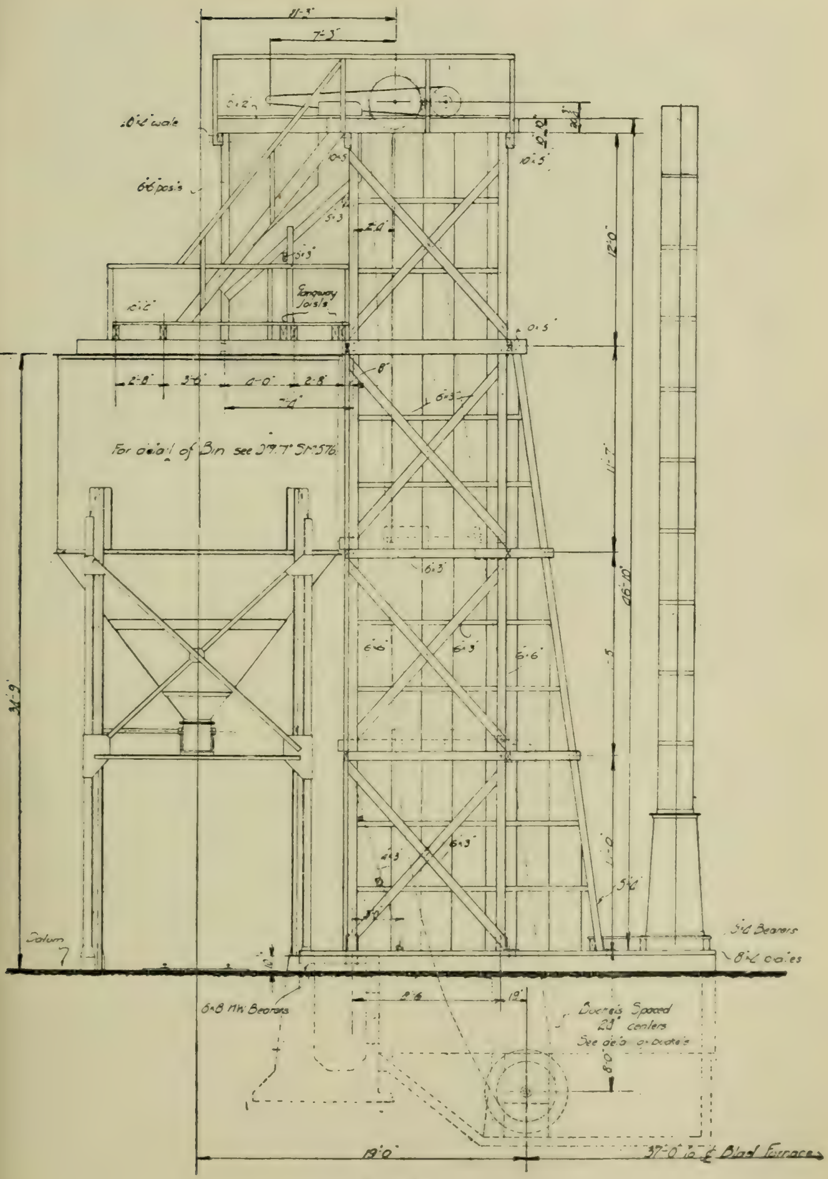


FIG. 21.
ELEVATOR AND STORAGE BIN.

elevator belt is ordinary 3-ply rubber conveyor belt running at a speed of 216 ft. per min. To this belt at 36-in. pitch are attached the buckets. The elevator buckets are 16¾ in. long x 8 in. wide x 6 in. deep, and have a capacity of 576 cub. in. The ends of the buckets are perforated, and the bottom has a drainage lip extending ⅝ in. from each end. By this means the water entrained by the slag filters out and is diverted clear of the belt and the lower buckets.

The elevator is driven by a 5-h.p. motor operating through a chain of gears. The speed of the motor is 940 r.p.m.

The elevator discharges into a 150-ton storage bin. The bin is constructed of ¼-in. steel plate throughout, and is a circular bin with a conical bottom. The cylindrical section of the bin is 16 ft. in diam. and 11 ft. 3 in. deep. The conical section on its larger diameter is 16 ft., and on the smaller diameter 16 in., with a vertical depth of 9 ft. 6 in.

The bin has a cast-iron undercut quadrant gate, operating on a delivery chute 16 in sq.

(*f*) *Disposal of Slag from Bin.*—The removal of the slag from the bin is done on day shift by means of a locomotive service. The track is 3 ft. 6 in. gauge, with 40-lb. rails, and runs from the bins to the reserve dump.

The locomotives are Barclay 6-coupled shunting engines, with outside valve link motion and steam brakes. The trucks are steel side-tipping cars, and hold, on an average, 7 tons of slag. Seven of these trucks comprise an average rake. The average time required to fill and dump a rake of trucks is 30 min., or approximately 100 tons of slag per hour. The labour on the locomotive service is one driver and one shunter, the latter operating the bin-discharging doors.

(*g*) *Dumped Slag.*—The granulated slag is an ideal material for handling on the dump. The slag dump is about 31 ft. high vertically, and the angle of repose is 35° from the vertical.

A typical sizing test on the granulated slag shows :—

On								Through
% ¼-in.	% ⅛-in.	% 10 mesh.	% 20 mesh.	% 30 mesh.	% 40 mesh.	% 50 mesh.	% 60 mesh.	% 60 mesh.
0.4	11.0	9.2	49.3	20.9	4.4	1.8	1.5	1.5

(*h*) *Granulated Slag to Blast Roasting Department.*—As previously indicated, granulated slag forms a constituent of the blast roasting charge. The slag delivered to this section is transported in side-tip trucks from the respective storage bins and tipped into a small bin, the top of which is level with the railway track. It is then transported to the blast roasting mixture bins by a 20 in. wide 4-ply rubber conveyor belt running at an inclination of 20½° from the horizontal and at a speed of 160 ft. per min. The average moisture in granulated slag ex storage bins is from 2.5 % to 3.5 %.

It is interesting to note that with slag of this sizing test and moisture the construction of the above-mentioned bin is such that it absolutely clears itself after each truck has been dumped.

In a bin of this type an essential consideration is the slope of the sides and the valley angle—that is, the angle formed in the corners where sides and ends meet. Experience has shown these angles to be :—

Slope of sides, from 42° to 45° from vertical.
Valley angle, 67½°.

SECTION K.

ACCESSORY FURNACE OPERATIONS—PROCEDURE IN BLOWING IN A FURNACE.

A wood fire is made in a previously-dried crucible of a furnace and kept burning for 48 hours prior to starting. This is necessary to ensure the brickwork of the crucible and the lead syphons being sufficiently hot to prevent the chilling of the bullion when the furnace is started.

The ash from the above fire is then removed and the melting-down fire made, which consists of a good fire of large, free-burning logs. Bars of bullion are then thrown on the fire and melted down until the crucible is full of molten bullion. The ash and charcoal are then scraped off the surface of the bullion and fresh wood is added to make a good lighting-up fire for the coke. The tapping breast and slag spouts are now built into position, and the furnace made ready for charging from the feed floor. The time required to melt down and fix up the tapping breast, etc., is usually about eight hours, and requires one tapper and four labourers for the operation.

The amount of bullion required to fill the crucible of a large furnace is from 900 to 1050 bars, or approximately 30 tons. Before commencing to charge from the feed floor, about 3 tons of dry wood of medium size is thrown in from the top, care being taken to get it evenly distributed in the furnace. It ignites instantly, and then charging is proceeded with from the charge cars as follows:—

1 charge of coke	4000 lb. coke.
12 charges of slag..	6000 lb. slag. 600 lb. coke. 4 bars bullion.
Fill up with the following charge (about 20 required)	2000 lb. H. & H. sinter. 2000 lb. slag. 400 lb. coke.

When the furnace is nearly full the blast is turned on to about 4 oz. at first. After a period of 10 to 15 min. the fire is examined through the tuyeres openings, and, if burning freely, the blast is increased to 10 oz. This is sufficient blast until slag begins to flow, usually in 1 to $1\frac{1}{2}$ hours. When slag and bullion are both running freely, and the top of charge moves uniformly down the furnace shaft, the ordinary full ore-charge is put on and blast gradually increased. At the end of 24 hours the furnace is usually on full blast and in good working order.

PROCEDURE IN RUNNING A FURNACE DOWN.

When blowing-out a furnace the following practice is adopted:—

Four slag charges, each consisting of 4000 lb. slag and 320 lb. coke, are given, and the furnace run down, gradually reducing the blast until the top of the charge is about 15 ft. below the feed-floor level. The furnace is then stopped, and the top accretions barred off in the usual manner. Two more slag charges are now dropped on top of the barrings, and the furnace re-started and continued until the slag ceases to run from the tap-holes. If the furnace, while running down, becomes very hot on top, additional slag charges are added.

During the latter stages of running-down the slag is allowed to run continuously from the tap-holes and a minimum blast of 6 oz. employed, since below this figure the slag becomes very thick.

The blast is now taken off, the tapping breasts removed, and a hole cut through the charge down into the crucible. This allows the bullion being baled in hand ladles and cast into the ordinary moulds arranged on a convenient staging.

The end jackets are now removed, and the furnace allowed to cool off, which usually requires a week. The water is then turned off the remaining jackets, and the furnace made ready for final chipping-out.

After running-down, the material left in the furnace should not be more than about 1 ft. above the tuyeres, and this material, together with the chippings from the shaft, is returned to the other furnaces for re-treatment.

PROCEDURE IN BARRING A FURNACE.

Prior to running a furnace down for barring, four slag charges, consisting of 4000 lb. slag and 320 lb. coke, are given. As the top of the charge recedes the blast is reduced to prevent the top of the furnace becoming too hot and to permit of barring off an appreciable amount of the top accretions while the furnace is going down. It is usual to run down until the top of the charge is about 2 ft. below the bottom of the accretions ; but this, however, is not always possible, since in cases the accretions extend right down to the tuyeres.

Effective barring cannot be carried out much below 15 ft., and this is the depth at which the furnace is usually stopped. Before barring is commenced, 1000 lb. of scrap iron and 500 lb. of coke are thrown in.

The barring is done by either Sullivan poppers, with 1¼-in. octagon steel with diamond point, or by hand, with bars 1½-in. octagon steel and 14-lb. hammers.

Whilst barring, coke is added from time to time, the amount depending on the condition of the furnace and the amount of accretions. A fair average quantity is 1200 lb. When barring is complete, four slag charges similar to the above are fed in, and then ordinary ore charges, until the top of the charge is within 3 ft. of the feed floor. The blast is turned on to about 20 oz. at first, and, when slag is running freely and the charge begins to sink evenly in the shaft, the blast is turned on to normal pressure and normal charging commenced.

Whilst the furnace is barring, a bar is driven through each slag tap-hole down into the lead in the crucible, thus ensuring a connection between tap-hole and lead syphon. The bars used are 1-in. octagon steel, curved at the end to give a down course. If the crust in the crucible is too hard to be treated in this manner, the tapping breasts are removed and connection made through to the lead syphons by means of gads and short bars.

PROCEDURE IN FREEING LEAD WELLS.

Whenever the syphon of the lead well becomes obstructed by incrustations, etc., it is customary to free these by the use of bombs.

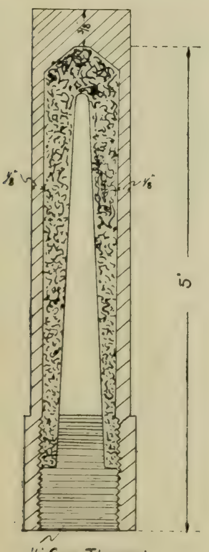


FIG. 22.
SMELTER'S BOMB.

The charge canister is as per Fig. 22. The rod to which the canister is attached is of ½ in. diam. irons, wedged out to give a butt 1½ in. long, which is threaded with ½-in. gas thread. Prior to charging the canister is lined with damp fireclay, and then moulded with a tapered steel rod having a rounded point and a shoulder, in order to keep the clay below the threaded portion. When dry, the charge of 10 grm. of dynamite is tamped in about ⅛ in., dry fireclay put on top, and then a cardboard wad.

The method of using the bomb is as follows:—The operator, wearing leather gloves and head mask, in case of a premature explosion, forces the bomb well down into the lead well. The heat of the lead causes the explosion of the charge a few seconds after immersion. The explosion of the bomb usually clears the obstruction in the syphon.

For a period, experimental trials were conducted with the canister filled with water instead of explosive. It was found, however, that the explosive force was much weaker, and not sufficiently strong to clear the syphons. These were discarded, and the dynamite bomb reverted to.

PROCEDURE IN REMOVING A JACKET.

This is a straightforward operation, and Fig. 23 of the appliance used for removing the jacket should suffice to illustrate the process.

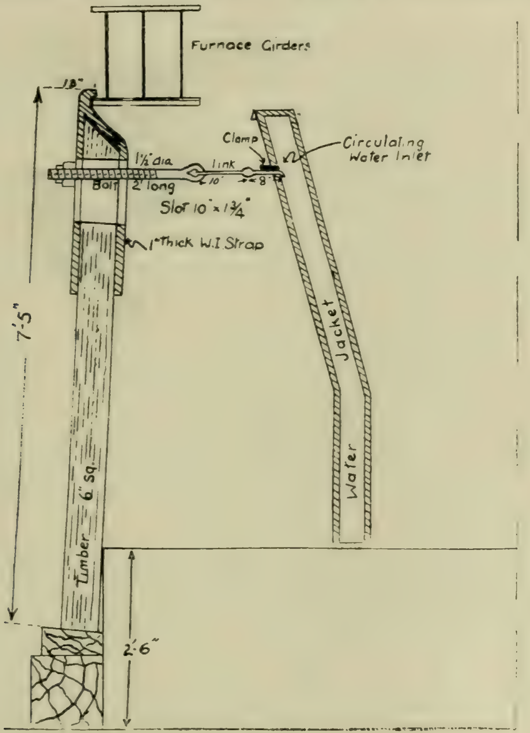


FIG. 23.

GEAR FOR REMOVING A JACKET.

The strut is made of 6-in. square timber, shod on one end with a W.I. strap 6 in. wide x 1 in. thick, and about 3 ft. over all. A hook is made at one end to fit over the flange of the main furnace girder. Through the end of the strut, which is shod, is a slot 1¾ in. wide x 10 in. long, passing through the shoe and timber. Through this slot passes a 1½in. diam. eye-bolt, 2 ft. long, with square thread. Attached to this bolt is a 10-in. link and a square-sectioned iron hook. This latter is placed in the circulating water inlet of the jacket and kept in place by a wedge clamp. By tightening up the nut on the end of the bolt sufficient power is obtained to remove the jacket from its old seating.

SECTION L.

FURNACE ACCRETIONS.

These accretions are usually formed just above the jackets, and extend at times almost to the feed floor. Their exact position, however, is to some extent controlled by the speed at which the furnace is operating.

In a fast-running furnace the accretions are generally high up in the shaft, the jackets and lower section of the brickwork shaft being comparatively free. In a slow-running furnace the accretions usually continue down to the jackets, and in some cases almost to the tuyeres.

Accretions are formed in two ways:—

(1) By volatilization in the lower regions and condensation in the upper or cooler part of the shaft.
(2) By portions of the charge fusing high up in the column (due to "hot tops") and adhering to the brickwork.

The rate of formation of wall accretions is influenced by the following considerations:—

(1) The presence in the charge of volatile compounds.
(2) The physical nature of the charge—*i.e.*, fineness.
(3) The position at which the coke is burnt.
(4) The rate of smelting.

The presence of large amounts of zinc in the blast furnace charge probably helps more than any other factor in the formation of accretions.

It is evident that some of the zinc compounds are reduced in the reduction zone of the furnace, and the metallic zinc travels up the charge and is oxidized in the upper zone of the furnace shaft and deposited on the charge in the cooler regions.

Excessive fines tend to produce a "hot top" furnace and slow smelting, and consequently the charge becomes pasty in the upper regions of the shaft. This condition rapidly produces accretions.

Under present conditions the furnaces are barred on an average about every 14 days, and the average time taken for each barring is 12 hours. The ordinary furnace crew is usually sufficient to enable this operation being completed in the above-mentioned time.

The following is an analysis of accretions recently formed in the furnaces. The variation in the composition of accretions covers such an extreme range that the analysis submitted cannot in any degree be taken as typical :—

	%
SiO_2	6.2
FeO	7.8
MnO	2.1
CaO	4.7
Al_2O_3	2.0
Zn	28.7
S	13.3
S	13.3 (no S as sulphate)
Cu	0.22
Pb	10.7
Ag	8.5
Ag	8.5 oz.

SECTION M.

TURBO-BLOWERS, BLAST DATA, ETC.

The blast for the furnaces is generated by turbo-blowers situated at the power plant.

There are three turbo-blowers :—

2 Fraser and Chalmers Rateau type.
1 Parsons high-pressure reaction.

The Fraser and Chalmers high-pressure Rateau type turbines, taking steam at 200 lb. pressure at the stop valve, are direct-connected to a two-stage blower. Each machine is capable of delivering 30,000 cub. ft. of free air per min. at a pressure of 5 lb. per sq. in. when running at a speed of 3200 r.p.m. The usual operating speed for furnace requirements is 2600 r.p.m., at a pressure of 44 to 46 oz.

The auxiliaries for these two blowers consist of Worthington rectangular surface condensers with Weymouth 10-in. centrifugal circulating pumps, direct driven by a 20-h.p. General Electric motor at 725 r.p.m., and Worthington two-crank vertical air pumps, gear-driven by 7-h.p. Phœnix motors.

The third turbo-blower is a Parsons high-pressure reaction turbine, direct-connected to a Parsons turbo-blower, with a capacity of 36,000 cub. ft. of free air per min. at a pressure of 4 lb. per sq. in. when operating at 3600 r.p.m.

The auxiliary equipment of this machine consists of an Allen condenser, Weymouth 10-in. centrifugal circulating pump direct-connected to a 20-h.p. General Electric motor, and an Allen 3-throw vertical air pump, gear-driven by a 7½-h.p. General Electric motor.

In order to minimize the chance of passing quantities of dust through the blowers with the incoming air, the inlets to the suction pipes are situated at the top of a set of towers.

The discharge from each blower passes through a hand-operated gate-valve through an automatic non-return and relief valve, and thence through a butterfly-valve to the 43-in. diam. steel main, which delivers the blast to the furnace department. The 43-in. diam. steel main passes behind the line of furnaces and between them and the main flue. From this main each furnace is served by a 27-in. diam. bustle pipe.

The air to the bustle pipe is controlled from the blast main by a butterfly valve, and to each bustle pipe is fitted a readily-operated relief valve in a position handy to the moulding floor.

AIR CONSUMPTION PER FURNACE.

A series of tests gave the following average air consumption per furnace when running at various smelting speeds.:—

No. Charges per 8-hr. Shift.	Blast Pressure.	Cub. ft. Air per min.
	oz.	
45	44–46	7400
40	44–46	6300
35	44–46	5400
No. 7 furnace on average running	24–28	2300

BLAST PRESSURE.

Under normal conditions the average pressure on the blast furnaces is from 44 to 48 oz. per sq. in.

SECTION N.

BAGHOUSE.

The gases for delivery to the baghouse are drawn from the main brick smelter-flue at a point about 50 ft. from the base of the main stack. Between the stack and the baghouse offtake the main flue is controlled by a damper, situated at the base of the stack. This arrangement allows of the whole or part of the furnace gases being discharged to atmosphere per medium of the stack, should it be so desired, while repairs, etc., are effected in the baghouse.

The gases are drawn from the main flue by a fan of the impeller type. The diameter of the fan is 7 ft., and the width of the blades is 6 ft. 6 in. The fan is driven by a 75 h.p. motor running at 460 r.p.m., belt-driving the fan at a speed of 350 r.p.m. The rating of the fan is 90,000 cub. ft. per min. against a pressure of $1\frac{3}{4}$ in. of water.

The flue connecting the fan to the main smelter flue is of steel, 6 ft. 6 in. sq. and 25 ft. long.

The fan delivers to a concrete flue, 7 ft. sq., which in turn connects with the main header flue of the baghouse.

The main header flue is of steel, and is 6 ft. sq. and 90 ft. long. This flue runs parallel to the northern end of the baghouse, and is supported on structural steel $12\frac{1}{2}$ ft. above ground level. It is so designed as to cope with the six chambers of the baghouse, but to date only three chambers have been equipped in the new installation.

Since the baghouse proper comprises a set of chambers which are identical in construction and fittings, the following notes will be confined to a description of the arrangement and fittings of one chamber.

DETAILS OF CHAMBER.

The walls of the chamber are of ordinary red brick, the roof being of timber covered with malthoid. The internal dimensions of each chamber are :—Length, 57 ft. 6 in. ; width, 14 ft. ; and height, 50 ft.

At a height of 11 ft. 2 in. above the ground floor of the chamber is built a wrought-iron plate floor extending over the whole area of the chamber. In this plate-floor are four parallel rows of 20-in. diam. holes, each hole being surmounted by a thimble, to which the bottom or open end of the filtering bag is attached. Fig. 24 shows the placing of these holes.

It will thus be seen that there are four rows of 20 bags, so placed as to give ample gangway room for inspection of or replacement of damaged bags.

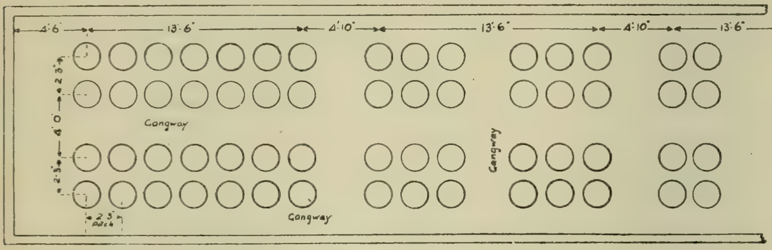


FIG. 24.
BAGHOUSE CHAMBER PLAN.

Fig. 24A shows details of the thimble, which is made of 1/16-in. W.I. plate. These thimbles are closely fitted to the W.I. floor-plates so that no leakage of gas can take place between the thimble and floor.

In each chamber there are two parallel groups of 40 bags separated by a central gangway. Immediately under each group (of two parallel rows of bags) a hopper of a V-shaped section is constructed on the other side of the plate floor. This hopper is gas-tight, and serves a dual purpose—

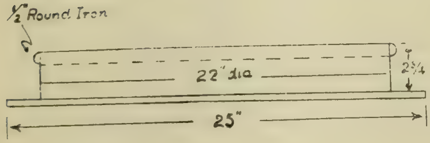


FIG. 24A.
THIMBLE.

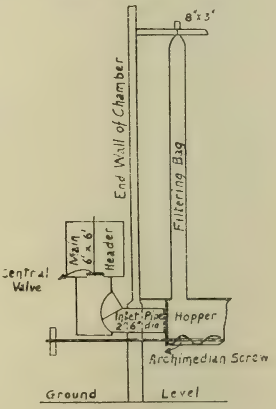


FIG. 25.
VALVE.

(*a*) Acting as a header to evenly distribute the gases to each group.

(*b*) Acting as a hopper to catch the fume when the bags are shaken.

At one end of this hopper the main gas inlet pipe, 2 ft. 6 in. in diam., enters, making two 2 ft. 6 in. diam. inlet pipes to each chamber of 80 bags. At the bottom of this hopper works an Archimedean screw, having a diameter of 9 in. and a pitch of 10 in., and travelling at a rate of 40 r.p.m. This delivers the fume to a truck outside the chamber. The discharge end is kept sealed, when not delivering, by a piece of flannel securely tied.

The passage of the gas from the main 6 ft. x 6 ft. steel header to each chamber is controlled by two valves, one of which is shown in Fig. 25.

Each chamber of the baghouse is a distinct unit, and can be operated entirely independently of the others.

METHOD OF HANGING BAGS.

Running transversely across the chamber, and at a height of 29 ft. 6 in. above the steel plate floor and centrally over each transverse line of 4 thimbles, are 8-in. x 3-in. timber beams. The top of each bag is fitted with two lugs (of flannel), which are attached to two hooks at 12-in. centres on the underside of the 8-in. x 3-in. beam. The bags then hang vertically over the thimble, and are tightly tied around this.

METHOD OF SHAKING BAGS.

The shaking movement of the bags is imparted by two 3/16-in. wire ropes passing through loops on the side of the bag. For a length of 18 ft. 9 in. these ropes are parallel to the side of the bag; at the bottom they splay out to a width of 3 ft. 9 in. in 6 ft. vertical height. Similarly on the top. The bottom section of the wires is attached to hooks set in the floor. at 3-ft. 9-in. centres, as indicated in Fig. 26. The upper section of the wires is attached to hooks fitted in the longitudinal beams of the shaking gear. The upper hooks are vertically above the lower hooks.

The shaking gear (see p. 195) has a vertical movement of 2 in. imparted to it by means of a cam travelling at 60 r.p.m.

On each stroke of the cam the two wires attached to the side of the bags straighten, thus imparting to the sides of the bag an inward and outward movement, and effectively dislodging the accumulated fume.

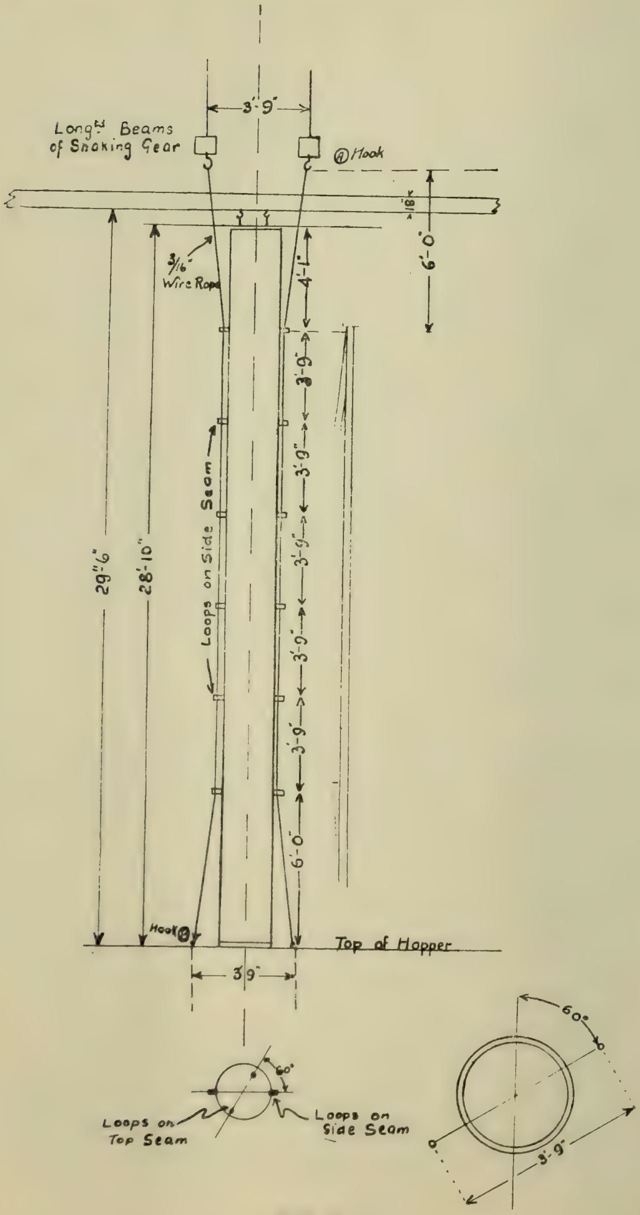


Fig. 26.

Shaking Gear.

SHAKING GEAR.

The plan of the chamber shows that longitudinally the parallel groups of 40 bags each are divided into three groups, two of 14 bags each and one of 12 bags. Over each of these respective groups is installed a separate shaking frame (shown in Fig. 27).

Running longitudinally under this shaking gear are two cam shafts running at 60 r.p.m. On these cam shafts are fixed three single-throw cams of 2-in. throw, which come in contact with the shaking frame by means of the two cam rollers, 2½ in. diam. x 2⅛ in. face. The throw of the cam compresses the springs, thus giving a quick return. The side wires of the bags are attached to the 4 in. x 2 in. timber beams. There is thus to each chamber two parallel rows of shaking gear and four cam shafts.

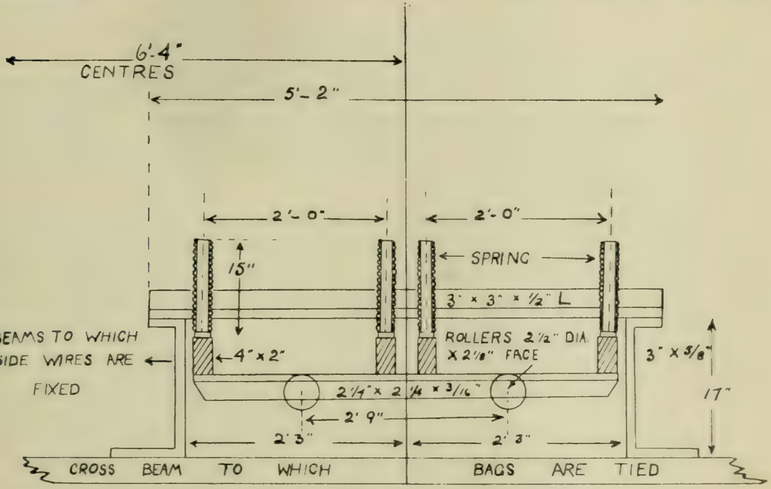


FIG. 27.

CROSS SECTION OF SHAKING GEAR.

MAKING OF BAGS.

The flannel of which the bags are made is obtained from the manufacturers, Messrs. Collins Bros., Geelong, Vic., in rolls of 70 ft. length and 72 in. wide. The sewing wool used is 2-ply. The machine used for making the bags is a Singer sewing machine, type 55-3.

The finished dimensions of the bags are—length, 29 ft. 6 in. ; diam., 22 in.

In making the bags the selvage edges are lapped 1 in. and doubly sewn. The folded side is similarly done. The top of the bag is lapped and sewn twice. The top lugs, made of flannel folded over, are 2 ft. 6 in. in length, and are attached to the bags by copper rivets and leather washers. The side lugs are 4 in. long, and are sewn to the bag. The location of these side lugs is given in sketch accompanying "Method of Shaking Bags."

BAGHOUSE OPERATIONS.

(*a*) *Operating Labour.*—Since the bags are shaken by mechanical means, as previously described, and the collected fume delivered by an Archimedean screw, one labourer per shift is employed to attend to the baghouse.

(*b*) *Gas Control.*—As previously indicated, the baghouse system is so designed that, if required, a portion or the whole of the furnace gases can be discharged to atmosphere through the main stack. This may be necessary during a temporary breakdown in the baghouse system or during an overhaul of the system.

Again, pending the completion of the baghouse—that is, in connection with the installation of mechanical shaking gear, etc., in the remaining three chambers—it has been found that the bag-filtering area is not capable of handling the whole of the furnace gases.

It is obvious that when a furnace is running down, preparatory to barring, an abnormal amount of false air is drawn into the flue system. Again, should the furnaces be running with "hot tops," the temperature of the gases may rise above the safe working limit of the baghouse. The maximum allowable safe-working temperature and pressure at the baghouse intake is—

Temperature	200° F.
Pressure	2½-in. water gauge.

(*c*) *Shaking of Bags.*—The bags are regularly shaken every hour, or at such periods as the resistance rises to the limit of 2½-in. water gauge. The material dislodged from the bags falls into a hopper-bottomed bin, and by means of a screw delivery is fed to side-tip trucks of about ½-ton capacity. As each truck is filled it is removed to a burning-off chamber and ignited on the surface of the fume by means of shavings. The fume ignites readily, and, at the end of about

4 hours, has decreased 50 % in volume, and is in a solid, compact state. It is then transported to the feed floor of the blast furnaces and fed along with the ordinary charge to the furnaces.

(*d*) *Fume Recovered.*—The lead obtained from baghouse material is approximately equal to 1.5 % of total new input lead to the blast furnaces.

Over a half-yearly period the analysis of burnt baghouse fume was :—

	%
Pb	67.4
Pb as $PbCl_2$	4.0
Ag	1.4 oz.
Cu	0.083
Zn	7.9
S	7.17
S as SO_4	2.77
Sb	0.25
As	0.48
Cn	2.12
Cl	2.32

SECTION O.

TREATMENT OF COPPER DROSS EX REFINERY.

The dross ex the copper-softening furnaces at the refinery is treated in No. 7 blast furnace in order to recover the major portion of the lead content of the dross, and at the same time produce a copper matte of a grade suitable for further treatment.

The following is an average analysis of the copper dross :—

Analysis of Cu Dross.

	%
Pb	72.20
Fe	0.90
Cu	10.83
As	2.47
Sb	0.83
S	5.10
Zn	2.81
Ag	30.0 oz.

The copper dross is delivered from the refinery and stacked in paddock bins on ground level at the smelters adjacent to No. 7 furnace.

The dross and scrap iron portion of the charge are delivered to the feed floor by means of hand-drawn charge barrows.

No. 7 furnace is hand-fed, and has an open top. The coke and sinter portions of the charge are delivered by the ordinary charge truck from their respective bins.

The charge on a copper dross campaign is as follows :—

	lb.
Copper dross	3000
D. & L. sinter	800
Scrap iron	220
Returned slag	1000
Coke	340

The blast pressure is usually about 20 oz.

The following figures apply to a fortnight's campaign on the above charge :—

Average No. of charges treated per 24 hours ..	67
,, ,, ,, bars bullion produced	2349
,, lead in slag	1.7 %
,, tons matte produced per 24 hours.. ..	15 tons.

The bullion produced carried a heavy dross, analysis of which gave, on three samples :—

	% Cu	% Pb	% Zn	% Fe	% S
No. 1 sample ..	32.7	33.4	2.5	7.5	5.4
No. 2 sample ..	43.6	27.9	2.3	5.4	5.1
No. 3 sample ..	44.0	31.7	2.5	2.2	2.8

The slag produced, together with the matte, is tapped into horse-drawn slag pots, allowed to chill in the pot, and delivered to the slag pit, where the matte is separated by hand-picking.

The average analysis of the copper matte now produced is approximately—

	%
Cu	33.5
Pb	18.3
Zn	6.4
Fe	14.5

SECTION P.

TREATMENT OF ANTIMONIAL SLAG.

The antimonial slag is produced from the refinery antimonial dross furnace when the skimmings ex antimony-softening furnace are reduced by carbon. This slag is treated in the No. 7 blast furnace, and is delivered to the furnace in the same manner as copper dross.

The following is the analysis of the antimonial slag. The analysis is made by reducing the metal in the slag by carbon and then anlayzing the reduced metal :—

Analysis of Antimony Slag.		Typical Antimonial Slag Analysis (Weekly).	
	%		%
Pb	56.88	Metal reduced	57.5
Sb	15.91	Containing—	
As	11.60	Sb	19.4
Cu	0.16	Cu	0.12
SiO_2	1.67	As	0.11
FeO	0.55	Ag	1.0 oz.

As previously mentioned, No. 7 furnace has two distinct flues—

(1) One which connects to the ordinary smelter flue system, so that the gases pass through the baghouse.

(2) One which connects with the smelter stack direct.

When treating antimonial slag the dampers are closed so that connection to the baghouse flue system is cut off and the furnace operates on the auxiliary flue leading direct to the main stack. This is done in order to keep the high arsenic content of the gases separate from the normal smelter gases, as otherwise the high arsenical fume would vitiate the normal baghouse material.

On one occasion the gases from this furnace were all filtered when the furnace was treating antimonial slag, and the following is an analysis of the baghouse material (raw) :—

	%		%
Insol.	0.14	S as SO_4	0.27
FeO	0.25	Cu	0.05
MnO	0.09	As	39.0
CaO	0.39	Sb	11.62
Al_2O_3	0.21	Cd	1.3
Zn	7.6	Pb	19.5
S	2.6	Pb as $PbCl_2$	20.4

Before swinging No. 7 furnace on to an antimonial slag charge, it is necessary to thoroughly wash out the crucible in order to eliminate loss of silver in the antimonial lead. General practice is to wash out the furnace for a period of 24 hours with low silver-bearing materials, such as burnt smelter baghouse material or dross ex the refining furnaces.

When the bullion so produced shows under 3 oz. per ton, the furnace is ready for charging with antimonial slag.

The charge is—

	lb.
Antimonial slag	1200
Returned pit slag	800
Scrap iron	300
Coke	280

Average blast pressure, 16 oz.

The products obtained are :—

(1) An antimonial lead, which requires drossing.

(2) Slag and speiss, which are tipped over the dump.

The following figures apply to a campaign conducted from 14th March, 1922, to 3rd April, 1922 :—

Average number of charges per 24 hours ..	70.
,, ,, ,, bars antimonial bullion ..	808.
,, arsenic in metal	0.19 %.
,, silver in metal	3.2 oz.
,, metal in slag	0.97 %.
Total antimonial slag treated	793.4 tons.
,, undrossed antimonial metal produced ..	432 tons.
,, slag dumped	831 tons.

The maximum arsenic allowable in the metal is 0.5 %, and to ensure a product below this figure it is necessary to maintain high reduction, slow speed, and use good solid scrap iron.

DROSSING OF CRUDE ANTIMONIAL METAL.

The crude antimonial metal is drossed in the shrapnel kettle adjacent to No. 7 furnace. This kettle is a replica of the refinery kettle. In the run previously mentioned the following figures apply to the drossing operation :—

	Tons	cwt.	qrs.	lb.
Average charge to drossing kettle	44	17	1	4
,, weight of dross removed per charge ..	14	4	3	8
,, ,, ,, drossed metal produced per charge	30	2	1	24

The kettle is charged and fired until the charge is melted. It is then cooled back until about a 2-in. ring of metal solidifies around the kettle. The dross is then removed and the clean antimonial lead moulded into bars weighing 104 lb. The total weight of drossed antimonial metal produced is 279 tons 13 cwt. 0 qr. 14 lb.

The metal is moulded into 10-ton lots, generally three lots being required to mould a kettle full. Before moulding, a sample of the metal is taken.

An average analysis of a typical charge before moulding is :—

Pb..	86.5 %.
Ag..	4.9 oz.
Cu	0.08 %
As	0.02 %
Sb	13.3 %.

The respective 10-ton lots, moulded from kettles, are branded "Lot 1A," "Lot 2A," "Lot 3A,"——"Lot 9A." Then "Lot 1B" ——"Lot 9B," and so on.

This means that three 10-ton lots (approximately) of the antimonial metal have the same analysis. The segregation into small 10-ton lots is to facilitate shipping without re-weighing.

A composite sample of the skimmings removed from the charge in the drossing kettle showed—

Pb..	68.6 %
Ag..	3.7 oz.
Cu	4.6 %
As	0.12 %
Sb	25.0 %

At present this skimming is returned to the refinery and worked off in a similar manner to hard lead from the dross furnace (antimonial).

The number of campaigns on the treatment of antimonial slag per year is entirely dependent on the demand for antimonial metal. The antimonial slag is stacked so long as stocks of metal are on hand.

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BLAST ROASTING PRACTICE IN THE TREATMENT OF LEAD ORES.

By P. J. Walsh.

Introduction.

In the present paper it is the writer's intention to largely confine the subject-matter to details relating to the metallurgical practice in connection with the blast roasting of lead ores at the Port Pirie works of the Broken Hill Associated Smelters Pty. Ltd. rather than to elaborate on mechanical plant details which are more or less standard equipment of the average Dwight and Lloyd or Huntington–Heberlein blast roasting installations (hereinafter referred to in the text as D. & L. and H. & H. respectively).

The present blast roasting plant is a combination of D. & L. straight line machines and 10-ton H. & H. pots. The latter installation was an operating unit prior to the introduction of the D. & L. machines, the pre-roasting then being accomplished in Ropp roasting furnaces.

On the introduction of the D. & L. machines the use of the Ropp roasting furnaces as a pre-roasting unit was discontinued, and this step of the process was then carried out on one section of the D. & L. installation.

Consequently, then, the present plant consists of—

(1) A section of D. & L. machines, which operate solely as a pre-roasting unit.

(2) A section of D. & L. machines which operate to their full capacity as a final roasting unit and treat the main portion of the pre-roasted material.

(3) A section of H. & H. pots, which operate as a final roasting unit and treat the balance of the pre-roasted material which the final roasting section of the D. & L. installation is unable to handle.

It is not within the scope of this article to discuss the relative merits of either D. & L. or H. & H. sinter as regards their suitability for treatment in the blast furnaces other than to show that, with our present practice, we can produce a finished product from either section of the plant carrying a low sulphur content.

In this article the subject-matter will be dealt with under the following sections :—

(1) Analyses and sizing tests of ores, fluxes, etc., treated in the process.
(2) Notes on and preparation of blast roasting charges.
(3) Pre-roasting.
(4) Intermediate crushing.
(5) Final roasting in
    (*a*) Dwight and Lloyd section.
    (*b*) Huntington–Heberlein section.

SECTION NO. 1.—ANALYSES AND SIZING TESTS OF ORES, FLUXES, ETC., TREATED IN THE PROCESS.

The major portion of the lead concentrates produced from the Broken Hill mines is treated at the Port Pirie works. The concentrates are of two classes—

(1) Granular concentrates—the product of jigs and Wilfley tables.
(2) Slime concentrates—the product of the flotation units.

In dealing with the matter of concentrates, typical analyses of the average grade of three of the larger mines are submitted, together with a sizing test on each class of concentrate.

ANALYSES OF CONCENTRATES.

	Granular Concentrates.			Slime Concentrates.		
	No. 1 Mine.	No. 2 Mine.	No. 3 Mine.	No. 1 Mine.	No. 2 Mine.	No. 3 Mine.
	%	%	%	%	%	%
SiO_2	2.8	8.8	8.25	6.45	3.3	8.05
FeO	3.75	4.4	3.60	5.41	5.7	5.47
MnO	1.76	5.4	2.35	1.41	1.8	1.30
CaO	0.95	1.0	0.80	2.63	0.34	0.42
MgO	0.10	0.31	0.14	0.27	0.09	0.45
Al_2O_3 ..	1.10	3.10	2.35	1.35	0.80	1.25
Zn	6.4	6.9	6.50	14.70	13.70	14.20
S	15.5	13.2	14.30	17.40	21.40	17.60
Cl	0.002	0.033	0.01	0.01	0.02	0.02
Cu	0.14	0.16	0.125	0.96	1.31	0.84
As	0.32	0.29	0.23	0.02	0.40	0.02
Sb	0.15	0.15	0.08	0.26	0.14	0.13
Pb	68.4	52.9	60.6	46.90	48.20	47.20
Ag. (oz.) ..	22.3	26.8	23.0	37.50	95.30	37.10
Au (gr.) ..	4.8	17.57	5.76	6.72	32.16	9.6

SIZING TESTS ON ABOVE CONCENTRATES.

	Granular Concentrates.			Slime Concentrates.		
	No. 1 Mine.	No. 2 Mine.	No. 3 Mine.	No. 1 Mine.	No. 2 Mine.	No. 3 Mine.
	%	%	%	%	%	%
On $\frac{1}{4}$-in. mesh ..	—	—	—	—	—	—
,, $\frac{1}{8}$-in. ,, ..	0.2	0.2	0.2	—	0.1	—
,, 10 ,, ..	—	0.1	—	—	—	—
,, 20 ,, ..	3.7	5.1	6.3	—	0.2	—
,, 30 ,, ..	8.5	6.2	9.3	—	—	—
,, 40 ,, ..	6.6	4.7	6.5	—	0.1	—
,, 50 ,, ..	9.2	6.5	8.4	—	0.1	0.1
,, 60 ,, ..	0.9	1.1	1.0	0.1	—	—
,, 80 ,, ..	12.6	10.5	9.6	0.1	0.4	0.1
,, 100 ,, ..	7.9	8.4	8.1	0.1	0.3	0.1
Through 100 ,, ..	50.4	57.0	50.5	99.4	98.5	99.3

(*b*) *Lime sand.*—This is a disintegrated and wind-blown deposit obtained on Wardang Island, which has, for reasons discussed in Section No. 3, "Pre-roasting," displaced crushed limestone on the charge.

The following is an analysis and sizing test typical of this material :—

Analysis.	%	Sizing Test.	%
SiO_2	5.7	On $\frac{1}{4}$-in. mesh	—
FeO	0.5	,, $\frac{1}{8}$-in. ,,	—
Al_2O_3	0.94	,, 10 ,,	0.1
CaO	47.6	,, 20 ,,	0.1
(= 85.0 % $CaCO_3$)		,, 30 ,,	0.2
MgO	3.4	,, 40 ,,	1.0
(= 7.14 % $MgCO_3$)		,, 50 ,,	7.2
Cl	0.052	,, 60 ,,	18.8
SO_3	0.085	,, 80 ,,	49.0
P_2O_5	0.16	,, 100 ,,	17.2
H_2O	0.16	Through 100 ,,	6.4
Organic matter	0.24		

(*c*) *Granulated Slag.*—This constituent of the charge is delivered to the blast roasting department direct from current production of the blast furnaces.

Typical Analysis.	%	Typical Sizing Test.	%
SiO_2	20.7	On $\frac{1}{2}$-in. mesh	0.1
FeO	28.0	,, $\frac{1}{4}$-in. ,,	0.2
MnO	4.5	,, $\frac{1}{8}$-in. ,,	11.0
CaO	11.6	Through $\frac{1}{8}$-in. ,,	88.7
MgO	2.0		
Al_2O_3	6.2		
ZnO	20.92		
S	2.7		

(*d*) *Silicious Ores.*—These ores are obtained from two sources—

(*a*) Carbonate ores from Broken Hill.

(*b*) Purchased ores from outside mines.

The following analysis can be taken as typical of carbonate ore :—

	%
SiO_2	43.9
FeO	7.75
MnO	2.7
CaO	0.8
MgO	0.36
Al_2O_3	7.0
Zn	0.80
Pb	25.0
S	0.8

(Ores purchased from outside sources are generally sulphide ores, and contain from 25 % to 30 % Pb.)

All silicious ores are crushed at the Port Pirie works to a fineness that 85 % of the material passes through $\frac{1}{4}$-in. mesh screen.

(*e*) *Risdon Residues.*—This name is applied to the residues obtained from the Electrolytic Zinc Co. of Australia, whose works are situated at Risdon, Tasmania. They contain Pb and Ag values which are not recovered in the leaching process. As regards slag-forming constituents, the following analysis is typical :—

	%
SiO_2	8.1
FeO	22.43
MnO	4.10
CaO	0.93
MgO	1.11

The sulphur in Risdon residues is mainly present as sulphate sulphur.

Section No. 2.—Notes on and Preparation of Blast Roasting Charges.

(A) GENERAL NOTES ON CHARGE.

It is interesting to compare the present-day blast roasting practice with the practice previously followed. The following comparison of past and present blast roasting charges constituents will indicate the line of difference :—

(1) Blast Roasting Charge—Past Practice:—
- (*a*) Granular concentrates.
- (*b*) Slime concentrates.
- (*c*) Silicious ore.
- (*d*) Crushed limestone.
- (*e*) Crushed ironstone.

(2) Blast Roasting Charge—Present Practice.—
- (*a*), (*b*), and (*c*) as above.
- (*d*) Lime sand replacing crushed limestone.
- (*e*) Ironstone deleted from charge.
- (*f*) Risdon residues.
- (*g*) Granulated slag.

Before discussing the steps which led to the adoption of our present practice, it is necessary to appreciate the essential factors which require consideration in the preparation of a charge for blast roasting ; they are :—

- (*a*) A thorough intermixing of the various components of the charge, so that every component enters with complete chemical combination with its neighbouring small particles.
- (*b*) A charge so constituted on thermal considerations that a sintering rather than a slagging reaction takes place.
- (*c*) A charge of such sizing that free passage of the blast is permitted equally over the whole of the bed.

Referring now to the sizing tests given in Section No. 1, it is seen that the major part of the charge comprises particles which will pass through ⅛-in. mesh. Consequently, then, it is evident that requirement (*a*) of the charge can be readily fulfilled.

This finer sizing of materials makes an interesting comparison with older practice. Then it was considered necessary, in order to open up the charge, that some of the components should be of large size. True, larger particles will open up the charge, but only in well-defined local areas, and as a result these areas give a freer passage to the blast and result in the major portion of the charge being short of air. Again, with coarse particles on the charge, experience showed that there was not that intimate mixture necessary to give a complete chemical combination, which is so essential. Consider the case of crushed limestone. Experimental work showed that the coarser particles did not completely enter into chemical combination with the mass of the charge. The particles were converted to CaO, which was carried in the charge as a mechanical rather than a chemical mixture. Consequently, these particles of CaO, when wetted or exposed to the air for any length of time, decomposed, causing the sinter to rapidly disintegrate. This materially defeated the object of the sintering process.

Again, a similar study of the crushed ironstone showed that the coarser particles were altered on the surface only, and, as regards the requirement of chemical combination, could be classed as practically inert.

The problem of crushing ironstone to such a degree of fineness as to overcome this undesirable condition presented difficulties on the score of expense. Consequently, efforts to delete it from the charge were made, and met with such success that for the past twelve months no ironstone has been added to the blast roasting charge.

Considering requirement (*b*) of the blast roasting charge, it is evident that if the heat of formation of the reacting charge components is too great, a slagging rather than a sintering action takes place. Consequently, there is the possibility of particles of sulphur being locked up in the slagged material of the pre-roasted material, with but little chance of being eliminated in the final roasting unit.

On this consideration it is necessary to add a diluent to the charge, and granulated slag appeared the remedy.

Granulated slag presents the following advantages:—

(1) It is preferable to crushed slag, being of much more regular size. With crushed slag the sizing scale varies from powder to coarse particles. Again, the presence of very finely divided slag, caused in crushing, results in a very fusible surface on the charge, which prevents free air passage. Consequently, then, granulated slag complies with requirement (*a*) of the charge.

(2) The granulated slag returned with the sinter to the blast roasting charge has value as regards slag-forming materials.

(3) Being one of the coarser materials on the charge, it assists in keeping the charge open.

(4) Owing to its fusibility it tends to strengthen the sinter by acting as a cementing agent.

Turning now to requirement (*c*) of the blast roasting charge, it is evident that in a very dry state the average sizing test on the whole charge would be too fine to allow good penetration of the blast. However, this disadvantage is overcome by careful control of the moistening of the charge, in which process a degree of nodulizing of the charge particles takes place. This moistening, then, has the following effect :—

(1) It causes the charge to be more nodular in condition, allowing good penetration of the blast.

(2) The conversion of the moisture into steam helps to counteract the difficulty of the heat of formation of the charge components tending to cause slagging.

Again, Risdon residues are of a nodular nature on account of contained moisture, and they, as well as containing lead and silver values, bring into the charge valuable slag-forming materials.

The next point to consider in connection with the charge is the mechanical preparation of the charge, in order to ensure that the charge components are thoroughly intermixed.

(B) PREPARATION OF CHARGE.

The various components of the charge are delivered from the stock piles or current railway-truck deliveries from Broken Hill to a series of bins known as the mixing plant.

Running transversely to this series of bins is a 20-in. conveyor belt, which acts as a charge assembly belt.

Under each storage bin of the respective charge components is an 18-in. conveyor belt which delivers each component of the charge to the main charge assembly belt. The delivery of these 18-in. conveyor belts is regulated by an adjustable door, so that the control of delivery of the respective charge components can be regulated in thin, fine limits.

From time to time the amount of material being delivered by these respective belts is determined by weighing their output over a period of time, and the rate of discharge adjusted as required. This arrangement then results in the charge components being evenly bedded on the charge assembly belt.

It is well to note that local requirements have necessitated the installation of a system of four conveyors between the charge assembly belt and the 15-ton supply hoppers to each machine. Each of these four belts runs at right angles to the other, so that before the charge has reached the conveyor, which distributes the charge to the pre-roasting machine hoppers, it has been subjected to the mixing given by its transference from the conveyor system at four points, and has passed through a bar cage revolving mixer before delivery to the main distributor belt.

From the main distributing conveyor the charge is given a further mixing by being transferred to the machine hoppers by means of a Robins traversing throw-off. At the discharge end of this transverse throw-off belt the charge passes through a revolving bar cage, built like a disintegrator, and is thoroughly mixed before being delivered to the machine hoppers. Consequently, then, to this stage the charge has been subjected to the following mixing :—

(1) Transferred to five conveyors.
(2) Finally passed through two revolving mixers of the Squirrel cage type.

The machine storage hoppers are of 15-ton capacity, and the charge from them is delivered to the machines by means of a further conveyor belt geared in direct ratio with the speed of the D. & L. machines.

In order that the charge may be evenly distributed across the machine pallets, the above conveyor delivers to oscillating chutes, which, being directly geared to the machine speed gear, traverse the pallets at a rate directly proportional to the speed at which the D. & L. machines are operating. This arrangement allows the charge being regularly delivered across the whole width of each pallet.

It is at the point where the charge is passing through this oscillating chute that the moistening of the charge is attended to. The matter of moistening will be dealt with in Section No. 3—" Pre-Roasting."

Again, in order to distribute the charge equally over the pallet, as regards depth, an adjustable levelling plate is fitted between the delivery of the oscillating chute and the dead plate of the machine.

The following are typical charges treated in the blast roasting process, together with the analyses of the sinter produced from each respective charge :—

Typical Blast Roasting Charges.

Charge No.	Coarse Concentrates.	Slime Concentrates.	Lime sand.	Silicious Ore.	" Risdon " Residue.	Granulated Slag.
	%	%	%	%	%	%
1	42	29	5	2	12	10
2	36	36	6	2	10	10
3	24	51	4	2	9	10
4	21	49	7	6	5	12
5	42	22	8	7	9	12

Analyses of Sinters.

Dwight & Lloyd—

Sinter from Charge No.	% Pb.	% SiO_2.	% FeO.	% MnO.	% CaO.	% Zn.	% S.
1	44.0	9.3	10.9	3.2	7.0	11.2	1.8
2	43.8	8.1	8.2	2.7	6.8	11.7	1.8
3	42.4	9.4	10.0	2.4	5.8	13.5	1.9
4	43.7	8.4	9.2	3.0	6.9	11.6	2.1
5	44.1	8.6	10.1	3.4	6.9	10.5	2.0

Huntington & Heberlein—

Sinter from Charge No.	% Pb.	% SiO_2.	% FeO.	% MnO.	% CaO.	% Zn.	% S.
1	43.7	9.5	9.9	3.1	6.9	10.5	1.9
2	43.4	9.0	9.0	2.9	6.6	11.3	1.6
3	42.5	9.6	9.5	2.5	5.6	13.1	1.6
4	43.2	8.9	9.0	2.7	6.8	11.2	1.8
5	44.5	8.6	10.0	3.4	6.8	9.8	1.9

SECTION NO. 3.—PRE-ROASTING OF THE CHARGE.

Before dealing with the subject of pre-roasting of the charge, there are certain factors of a mechanical nature in connection with the standard D. & L. straight line machines which are of interest to note. They are as follows :—

(1) *Vacuum Chambers.*—The total superficial area of vacuum chamber in each machine is 69.52 sq. ft.

(2) *Data re Fan.*—Each machine is equipped with a double-inlet direct-coupled impeller type of fan, of which the following are the leading dimensions and data :—

H.P. of motor-driving fan ..	75.
How coupled	Direct.
Speed of fan	970 r.p.m.
Area of inlet to fan	4 sq. ft.
Area of outlet from fan ..	6 sq. ft.
Diameter of impeller	5 ft. 6 in.
Width of impeller at periphery	10 in.
,, ,, ,, at centre ..	21 in.
Average volume of gas handled	12,000 cub. ft. per min.
Gases from fan delivered to ..	9 ft. diam. Cyclone dust collector.
Temperature of gas	127°C.
% SO_2 in gases	1.3 %.
Average vacuum	13–14 in. H_2O.

(3) *Type of Pallet Grate.*—On the pre-roasting section the machines are equipped with standard type Trail grates.

The grate is composed of 7 sections, four of which are fixed and three movable. Each section of grate intermeshes with its neighbour, the fixed and movable grate alternating.

On each side of the pallet proper a 2½-in. x ¾-in. angle-iron bolted to the side of the pallet keeps the grate sections in position. The fixed grate sections are, as well, attached to this angle-iron, while over the movable grate sections there is a slot in the angle-iron ¼ in. deep. This permits the movable grate sections a play of ¼ in., and by this means they are able to clear themselves of entangled charge.

As indicated previously, the correct moistening of the charge is an essential factor towards successful pre-roasting. About 7 % moisture is the amount aimed at to ensure good ignition.

If the charge is too wet it will not ignite, or perhaps ignite only on the surface, and the heat be not sufficient to carry the fire through. Again, if the charge be too dry it cannot be ignited, since it just lies dead on the pallet, with a tendency to form blow-holes.

The moisture for the charge is added just as the feed conveyor discharges into the oscillating distributing chute.

At this stage it is of interest to note the method employed in order to ensure that the charge is correctly laid on the machine pallets.

The oscillating chute distributes the charge evenly across the width of the pallet. However, it is not allowed to discharge direct on to the pallet, but delivers the charge across an inclined deflecting plate from which the charge rills to the pallets.

The advantages of this system are :—

(1) The inclined deflecting plate takes the impact of the falling charge, and thus counteracts the tendency of the falling stream of charge to cause a densely-packed charge on the pallet.

(2) Since the charge rills off this inclined plate, the larger nodules roll down to the toe of the rill, and thus form a uniform bed on the grate, preventing clogging of the grate.

The charge, having now been evenly and lightly distributed over the pallet, passes under an adjustable levelling plate, by which the depth of the bed is regulated.

There is a general tendency for the line of least resistance to the blast to be at the junction of the charge with the side plates of the pallets. To overcome this, the levelling plate is so designed that the charge for about 2 in. inward from the pallet side is about 1 in. deeper than the rest of the bed. Prior to this arrangement considerable trouble was caused by the formation of blow-holes on the sides of the pallets, with the result that the major portion of the bed failed to have sufficient air passing through.

Again, in this connection we found the design of the Trail grate faulty in the matter of the $2\frac{1}{2}$-in. x $\frac{3}{4}$-in. angle-iron locking piece on each side of the pallet.

These angle-irons made a butt joint with the angle-irons of the pallets before and after it. Consequently, then, when operating on a charge with a depth of bed of $3\frac{1}{2}$ in., it can be seen that at the junction of each pallet with the other a film of charge 1 in. deep x $\frac{3}{4}$ in. wide was the only security between the formation of a blow-hole at the pallet junctions on the sides. By bevelling off the ends of the $2\frac{1}{2}$ in. x $\frac{3}{4}$-in. angle-iron locking piece we successfully overcame the tendency of the charge to develop blow-holes at these points.

After passing the levelling plate the charge passes under the ignition stove.

The design of the ignition stove is illustrated in Fig. No. 1. It is essentially a reverberatory type of furnace, the throat of which is reduced to an area 3 ft. 7 in. wide by 8 in. long at the point where the heat is transferred to the charge. A coke fire is used in the ignition stove so that we obtain a clear gas, preventing the charge being coated with carbon, as would be the case if coal were used. Again, experience with this class of ignition has shown that, with the stove properly designed, perfect ignition is obtained by the heat radiated from the brickwork of the reverberatory area.

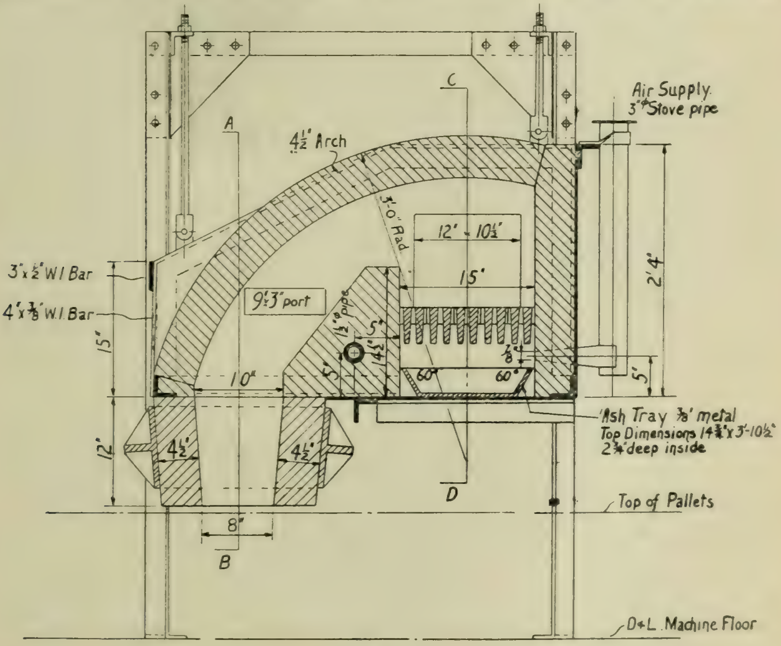


FIG. 1.
IGNITION STOVE.

It is not necessary in this paper to discuss the various types of ignition which have been employed from time to time at the Port Pirie works. However, the present system gives perfect ignition with a consumption over half-yearly periods of 13.2 lb. of coke per ton of charge ignited.

The ignition area of the stove is set immediately in front of the dead plate at the feed end of the machine.

Given the conditions previously stressed in connection with the mixing of the charge and the delivery of it to the pallets, and followed by good ignition, the pre-roasting process becomes practically automatic.

In the pre-roasting section of the plant the machine drive is so arranged as to give a variation in speed from 14 in. to 33 in. per min. In normal practice the general speed of the pallet travel is from 28 in. to 33 in.

As regards the sulphur content of the pre-roasted material, experience has shown that if this is around 6.5 % as a minimum and 8 % as a maximum a good final roasting is assured.

SECTION No. 4.—INTERMEDIATE CRUSHING.

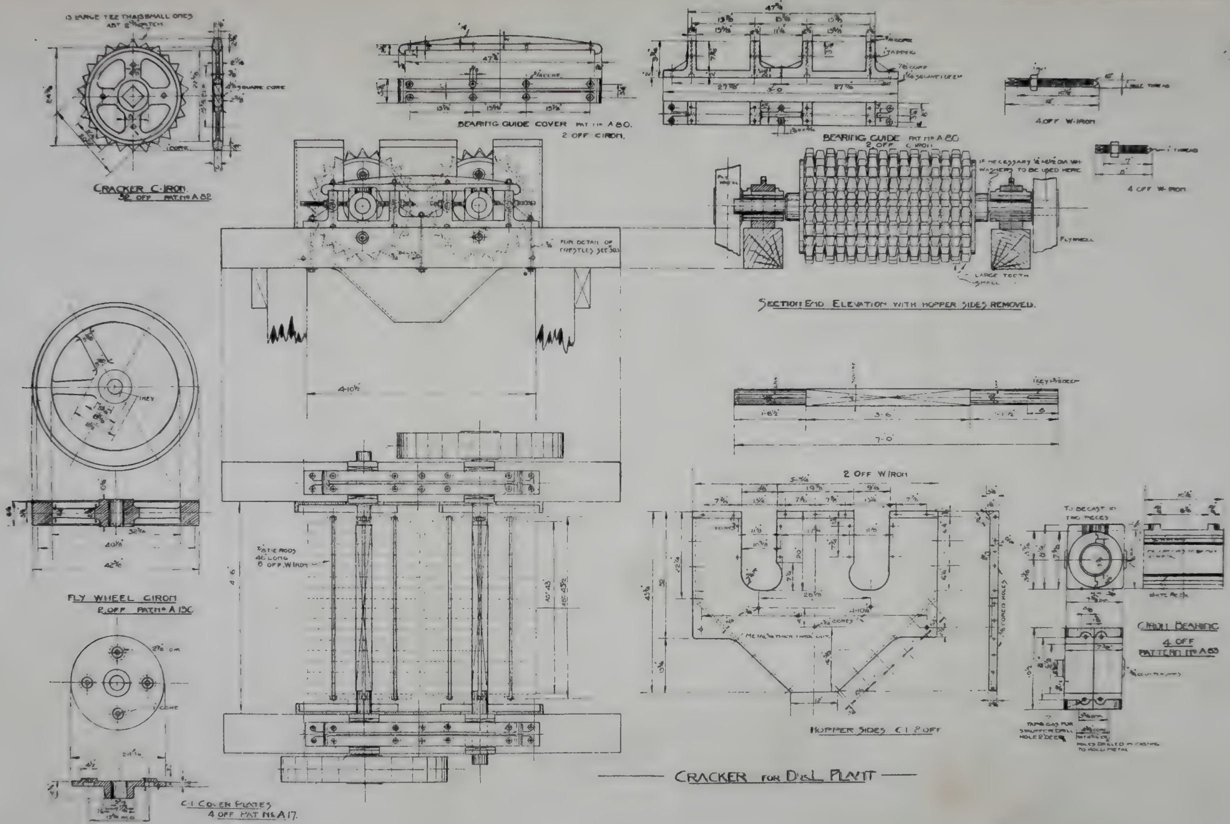
The pre-roasted material is discharged from the D. & L. machines on to a grizzly with bars at 6-in. centres. The fall on to the grizzly breaks up the material into a size convenient for conveying.

The first section of the conveying system by which the pre-roasted material is delivered to the cracker rolls consists of a 50-ft. length of steel-tray conveyor travelling at the rate of about 37 ft. per min. The average mean travel of the material from all machines on the steel-tray conveyor is about 30 ft.

This steel conveyor then delivers on to an ordinary 30-in. wide 6-ply rubber conveyor belt travelling at a speed of 227 ft. per min., and is eventually delivered to the storage hopper of the cracker rolls.

CRACKER ROLLS.

These are ordinary toothed intermeshing rolls, details of which are shown in Fig. 2. They run at 90 r.p.m., and crush the pre-roasted material from a 2-in. diam. ring downwards.



CRACKER FOR D&L PLANT

The coarsely-crushed material is delivered to a conveyor belt, which, in turn, delivers the material to the main crushing rolls.

MAIN CRUSHING ROLLS.

These are ordinary type smooth-face crushing rolls 3 ft. 9 in. in diam. x 2 ft. 6 in. wide, running at 60 r.p.m. They are so adjusted as to ensure that 85 % of the finally crushed product will pass through ¼-in. mesh screen.

The crushing of the pre-roasted material so that 85 % will pass through ¼-in. mesh screen is of vital importance in connection with the final sintering operation. The above gauge is one obtained on the result of experience, and it must be clearly understood that the minus ¼-in. mesh must not be too fine. Working on the above gauge, it has been found in practice that the minus ¼-in. mesh material complies with practical requirements.

If the material is crushed too fine it packs on the grates, and as soon as the fire comes in contact with it a rapid fusion takes place, and the fused mass forms a seal over the surface of the charge. This prevents penetration of the blast, and the result is a crust of semi-sintered material and a layer of raw or baked charge underneath.

Again, if the material is too coarse (say only 78 % passing through ¼-in. mesh) the charge bed is very porous, especially at the edges, and irregular sintering takes place.

SECTION NO. 5.—FINAL ROASTING.

(*a*) DWIGHT & LLOYD MACHINES.

The final roasting unit of the Dwight & Lloyd machines is a replica of the pre-roasting unit.

In this section great care must be exercised in obtaining a correct moisture of the charge in order that good ignition may be the result. This is of greater importance in this section than in the pre-roasting section, since the product of these machines is a finished product.

The speed of the final roasting machines varies from 17 in. to 22 in. per min. In this section the speed of the machines is so controlled that the material discharged from them is cold, and shows no sign of red heat. Hot-tipped material indicates unsintered material, with consequent high sulphur content and weak physical structure.

Considering the action of the various constituents of the charge on the quality of the sinter produced, practice has shown that most of the faults as regards sulphur content and physical structure can be controlled by adjustment of the lead, silica, or lime of the charge. Silica and lime have the most effect on the quality of the sinter.

Generally speaking, a high SiO_2 content (10 %) will give a strong cellular sinter high in sulphur. If much over 10 % the sinter is very fusible on the surface during the final roasting. This reduces the volume of blast penetrating the charge, and sintering takes place at a slow speed, and finishes as a hot mass with raw patches.

Lime in excess generally causes the lead to precipitate freely from the charge. This is a distinct disadvantage, since the reduced lead falls into the vacuum chamber or builds on the pallet rails, causing air leaks under the pallets. The sulphur content of the sinter, high in lime, is generally low, provided there is not also a high percentage of SiO_2 present.

If reference is made to Section No. 3 it will be found that the analysis of sinter produced from standard typical charges shows the relative percentages of SiO_2 and CaO in the sinter, which, as regards low sulphur content and strong physical structure, can be classed as exceptionally good.

(*b*) HUNTINGTON–HEBERLEIN POT SECTION.

The excess pre-roasted material from the D. & L. machines is treated in this section.

In Section No. 3 it will be seen that in analysis the H. & H. sinter compares favourably with the D. & L. sinter. The sulphur in both sinters shows very slight variation, and it is safe to state that the exceptional sinter now produced in the H. & H. pots at the Port Pirie works is the result of the class of pre-roasted material used. The coarser D. & L. pre-roast is a much more ideal charge to a pot than the fine pre-roast previously produced from the Ropp furnaces.

In pot sintering a much heavier moisture is necessary than on the D. & L. machines.

In the H. & H. sintering ignition is of vital importance. A bed of zinc distillation retort residues is spread over the grate to keep the bottom porous. Then a layer of hot coke, of a size that the maximum-sized pieces will pass $1\frac{1}{2}$-in. dia. ring, is spread uniformly over the layer of residues. This is followed by two trucks (30 cwt.) of material of a moisture content of about 4 %, tipped in such a way as not to disturb the bed of coke. This charge is spread evenly over the pot, and about 4 oz. blast used in order to keep the fire going while the balance of the pot is being filled.

The rest of the charge carries a heavy moisture of about 9 %, but a fair range of latitude is given.

When completely charged the surface of the charge is so adjusted that the sides are about 18 inches above the centre. This is done to bring the air through the middle first, as the tendency is for it to creep up along the sides of the pot.

The blast is then increased to about 10 oz., care being taken to blank any blow-holes and bring the fire up uniformly over the surface of the charge. When the fire is well up the blast is increased to about 20 oz., care being taken all the time to prevent blow-holes. As the sintering progresses the charge subsides, and the banks from the sides of the pot are gradually worked towards the centre.

With good ignition and careful attention the percentage of "fines" to be returned should not be greater than 12 %.

THE PRACTICE OF DE-SILVERIZING BASE BULLION AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

BY W. H. CAMPBELL.

THE de-silverizing of base bullion at the Port Pirie works is carried out in accordance with the principles of the well-known Parkes process.

The base bullion produced by the blast furnace department carries (apart from silver values) impurities, the chief of which are copper, antimony, and arsenic.

In order to more clearly indicate the grade of the base bullion, the following analysis, representative of a half-year's production of base bullion, is submitted :—

Ag	68.6 oz.
Au	0.096 oz.
Cu	0.86 %
Sb	0.67 %
As	0.32 %

Before proceeding with the actual operation of de-silverizing, it is necessary that the impurities, copper, antimony, and arsenic, should be eliminated. This is accomplished in the following steps :—

(1) *Removal of Copper.*— Copper is largely present in the base bullion as dissolved sulphide, and this is removed by liquation in a reverberatory type furnace, of which the following are the essential details :—

Hearth area	220 sq. ft.
Grate area	31 sq. ft.
Ratio hearth area to grate area	7 : 1.
Capacity	57 tons.
Working temperature	750° to 850° C.
Coal consumption per 24 hours	2 tons.

The hearth of the furnace is encased in a steel tank 22 ft. 4 in. long by 13 ft. 4 in. wide by 2 ft. 10 in. deep ; the bottom of the tank is covered by a depth of composition conforming to the invert :—

Cement	39 cwt.
Brick dust	8.4 tons.
Fireclay	0.6 tons.
Sand	0.6 tons.

When the bullion is melted in this drossing furnace a quantity of sulphide dross, amounting to approximately 8 % of the weight of the charge, floats to the surface. To free this dross from entangled lead it is necessary to fire the charge to about 800° C. At this temperature it is probable that a roast reaction is set up, liberating some metallic copper, which becomes alloyed with the bullion. The method of eliminating this metallic copper will be discussed at a later stage.

The dross now remaining on the surface of the charge is skimmed off and eventually treated in a small blast furnace, when the greater part of the entangled lead is recovered, while a Cu–Pb matte is produced. By this means the copper is eliminated from the circuit.

A normal time-table of the copper drossing furnace is as follows :—

	Hours.
Charging	$4\frac{1}{2}$
Firing	3
Skimming	3
Tapping charge to antimony softening furnace	$\frac{3}{4}$

A typical analysis of the copper dross obtained from this furnace is as under :—

	%
Pb	72.2
Fe	0.9
Cu	10.83
As	2.47
Sb	0.83
S	5.10
Zn	2.81

The bullion, after copper dressing, is of the following typical analysis :—

Cu	0.224 %
Sb	0.64 %
As	0.325 %
Fe	0.0033 %
Zn	0.0053 %
S	0.0235 %
Ag	66.7 oz.
Au	0.125 oz.

(2) *Removal of Antimony and Arsenic.*—The furnace used for the removal of antimony and arsenic is a replica of the copper-drossing

furnace, except that a water girder is placed in at the lead level, with a notch in the end girder, by which the dross is run off the charge. The working temperature of the antimony-softening furnace is from 1000° to 1100° C. This furnace is filled by gravitation from the copper-drossing furnace.

The normal cycle of operations on this furnace is :—

Charging	$\frac{1}{2}$ hour.
Firing	8 hours.
Cooling back	3 ,,
Tapping to zinc pans ..	$\frac{1}{2}$ hour.

The products from this furnace are—

(1) Antimonial dross—about 2 tons from each charge.

(2) Softened bullion.

And the following are typical analyses of the respective products :—

ANTIMONIAL DROSS.	
Pb	71.5 %
Cu	0.100 %
Sb	8.9 %
As	8.3 %
Fe	0.2 %
Zn	0.12 %
Ag	0.5 oz.
Au	0.6 gr.

SOFTENED BULLION.	
Cu	0.13 %
Sb	0.283 %
As	0.015 %
Ag	70.0 oz.
Au	0.15 oz.

In this furnace the antimonial and arsenical skimmings flow continuously through the notch in the end water girder, and are eventually treated in the antimony dross furnace. This furnace produces—

(*a*) Hard lead.

(*b*) Antimonial slag.

The former product is re-treated in the antimony softening furnace, being slowly worked off with the current softened bullion, while the latter is treated in a small blast furnace producing antimonial lead and waste slag.

As indicated above, the normal cycle of softening a charge is 12 hours.

In the softening process the arsenic is removed at a much faster rate than is the antimony. This is illustrated by the following experimennt, in which the antimony and arsenic content of the bullion was determined before firing and at intervals of 6 hours until the charge was clean. :—

	Sb %	As %
Assay of bullion before firing	2.22	0.32
,, ,, after 6 hours' firing ..	2.01	0.21
,, ,, ,, 12 ,, ,, ..	1.798	0.12
,, ,, ,, 18 ,, ,, ..	1.515	0.11
,, ,, ,, 24 ,, ,, ..	1.107	0.04
,, ,, ,, 30 ,, ,, ..	0.851	0.01
,, ,, ,, 36 ,, ,, ..	0.625	0.007
,, ,, ,, 42 ,, ,, ..	0.297	0.0007
,, ,, ,, 48 ,, ,, ..	0.157	nil
,, ,, ,, 54 ,, ,, ..	0.03	nil

Provided arsenic is not present to the extent of several per cent., it is readily eliminated in the softening furnace, being a readily-oxidized metal.

Consider now the effect of antimony and arsenic in the softened bullion in connection with the de-silverizing operation.

Antimony may be present to the extent of 0.5 % without seriously interfering, but 1.0 % has been found to have the effect of reducing the quantity of silver removed by a given quantity of zinc.

Arsenic, if present to the extent of a few tenths of a per cent., obscures the line of demarcation between the zinc crusts and the underlying bullion, owing to the arsenic entering the crust and making a mushy crust.

It will be shown later that the cycle of the zincing kettle is 24 hours, so on this account 2 zincing kettles are installed to each antimony softening furnace.

ZINCING KETTLES.

The zincing kettles are of hemispherical section, being 10 ft. 8 in. diam. internally by 3 ft. 7 in. deep at the centre, and are made of cast-iron, 1¾ in. thick throughout. The kettle is cast with a flange 6 in. wide by 1¾ in. thick, which serves as the support of the pan in the brickwork setting. The setting of the kettle is of freestone and brickwork surmounted by an annular ring of iron on the brick wall which supports the kettle by the flange.

These kettles have a capacity of 57 tons of bullion. They are direct-fired, having a grate area of 17½ sq. ft., and consume about 25 cwt. of coal per 24 hours.

The life of the zincing kettles depends to some extent on the method of firing, but to a very large degree on the quality of the cast-iron from which the kettles are manufactured.

It will be shown later that in the actual zincing operation the 57-ton charge of bullion has to be cooled back to solidification point four times and raised to approximately 450° C. three times in 24 hours. Consequently, rapid firing and cooling are essential if the cycle is to be run to time.

In the writer's experience, the average life of zincing kettles was increased from 8 to 13 months by care being exercised in the grade of iron from which the kettles were cast.

CYCLE OF OPERATION IN DE-SILVERIZING.

As mentioned previously, there are two zincing kettles to each antimony softening furnace, and for convenience in outlining the cycle these kettles will be called " A " and " B."

As soon as kettle " A " is charged by gravitation from the softening furnace a quantity of dross, which forms on the surface of the lead, is skimmed off. This dross is returned to the copper-drossing furnace.

The charge is now cooled back to the solidification point, and the Pb–Cu alloy, or cold dross, referred to under " Removal of Copper," is removed and likewise returned to the copper-drossing furnace.

A typical analysis of this dross is as follows :—

Cu	3.79 %.
As	0.86 %.
Sb	1.27 %.
Zn	0.30 %.
Fe	0.007 %.
Ag	68.4 oz.
Au	0.25 oz.

This " cold dross " is added to the copper-drossing furnace when the latter is nearly charged and covered with a sulphide dross. The metallic copper then combines with the sulphur and passes off with the copper dross from the furnace.

The following analysis is typical of the bullion after the removal of the "hot" and "cold" dross :—

Cu	0.077 %.
As	trace.
Sb	0.276 %.
Zn	trace.
Fe	0.007 %.
Ag	70.7 oz.
Au	0.15 oz.

The average time taken from the commencement of charging the kettles to the removal of the cold dross is 4½ hours.

The kettle is now strongly fired, and 325 lb. of zinc are added to the charge, and when the zinc is melted the whole charge is mechanically stirred thoroughly for 30 minutes. From the finish of cold drossing to the completion of the stirring-in process 2 hours are allowed.

This first addition of zinc (325 lb.) is known as "gold zincing."

Owing to the small amount of gold present in the bullion it is necessary, from the point of view of eventually recovering this gold, to obtain in the first or gold-zincing the maximum of gold and the minimum of silver.

This first or gold zincing takes up as well any copper which remained in the bullion after hot and cold drossing.

After the charge has been thoroughly stirred, the fire is drawn and the charge allowed to cool back. In this operation a crust is formed containing gold, copper, silver, and zinc. This is skimmed off with perforated skimmers, and the operation is so timed that the charge is just beginning to solidify when the final crust has been removed.

The time taken from the completion of the stirring in of the zinc to the removal of the gold crust is 4 hours.

The gold crust above referred to, together with 325 lb. of zinc, is now returned to kettle "B" as a gold-zincing.

Kettle "B" has been charged from the antimony softening furnace 12 hours after kettle "A."

The operation on kettle "B" is identical with that described in connection with kettle "A."

Again, the gold crust from kettle "B," together with 325 lb. zinc, is returned as the first or gold zincing to the next charge delivered to kettle "A."

The gold crust from this new charge in kettle "A" is now, together with 325 lb. zinc, returned as the first or gold zincing to the new charge in kettle "B."

The object of this four-cycle operation of the gold-zincing is to concentrate the gold from four charges of bullion into one crust.

After the fourth cycle the gold crust is pressed in the Howard press in order to remove the entangled lead and ensure a minimum tonnage of gold crust, which has to be further treated.

After the gold-zincing has been completed the temperature of the charge is raised to a degree sufficient to melt zinc and the first silver-zincing given.

While the kettle is thus firing, the crusts from a previous second silver-zincing, amounting to approximately 5 tons, are added to the charge. As soon as the temperature necessary to melt zinc is reached, the charge is mechanically stirred for 30 minutes. The crust from a second silver-zincing, referred to above, will be explained later.

After the stirring is completed the fire is drawn, and the charge allowed to cool back.

The crust which forms is removed by the Howard press, and the operation is so timed that when the final crust is removed the bullion just begins to solidify.

During the pressing of the crust the sides of the kettle are carefully scraped down in order to remove any alloy or crust adhering to the kettle.

When the available crust has been removed, that portion of the charge which has solidified around the edge of the kettle is pushed off into the molten bullion. Upon melting it will give up any alloy which it may have contained. This alloy is skimmed off by hand into moulds and added to the next kettle before the next silver-zincing.

The pressed crust referred to above carries approximately 1850 oz. per ton, and is now retorted and cupelled, producing silver of 998.5 fineness.

After the removal of this silver crust the charge is again brought to a temperature at which zinc will readily melt, and a second silver-zincing of 1100 lb. zinc given.

It is essential that the silver in the charge after the second zincing must not exceed 4 dwt. Ag per ton. Should the charge contain more than 4 dwt. Ag per ton an extra zincing must be given. This takes

6 hours, and, consequently, should this extra zincing be required, it is obvious that the rest of the unit is at a standstill for that period. Under these conditions it is necessary to add zinc in such quantities as to minimize the chance of a miss. This practice also provides so much available zinc in the final crust that, when used as a first silver-zincing on the next charge, the bullion will be reduced to 6–10 oz. silver per ton.

After the 1100 lb. of zinc has been stirred in, the charge is taken right back to the solidification point, and extra care is taken to ensure the last trace of alloy being removed, the kettle being scraped down and skimmed at frequent intervals. The charge is now heated up, and during the heating process the kettle is again scraped down to remove the last trace of crust.

A silver determination is now made on the charge, and if the result is within the 4 dwt. limit of silver the charge is dropped to the refining furnace. At this stage the lead contains 0.6 % zinc, this being the Pb–Zn eutectic.

In the refining furnace zinc and traces of other impurities are oxidized, forming a dross, which is skimmed from the surface of the bath of lead. A typical sample of this dross shows :—

Pb	83.1 %.
Sb	3.1 %.
Zn	13.7 %.
Ag	0.5 oz.

The zinc remaining in the lead after de-silverizing is wholly lost, and amounts to 13.4 lb. zinc per ton of lead. This loss is in no way dependent on the amount of silver in the bullion being treated, being a constant loss whether the silver be 1 oz. per ton or 100 oz. per ton.

A further quantity of zinc is lost in the retorting of the silver crusts, the average recovery being around 75 % of the total zinc held in silver crusts.

In the case of the Port Pirie practice a further zinc loss is sustained in the scorifying of gold crusts, since the gold in the bullion is so small as to require selective gold-zincing, and prevents a first gold–silver crust being obtained in which the ratio of Au to Ag would be such as to permit the final dross from these crusts being economically parted.

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THE TECHNICAL ANALYSIS OF LEAD BLAST FURNACE SLAG, AS PRACTISED AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

By J. E. P. Murrie.

The slag from the blast furnaces having been granulated, a representative sample is taken to the sample mill. It is then dried, crushed through a 100-mesh screen, intimately mixed, and packeted for assay purposes.

Method of Analysis.

INSOLUBLE.

Weigh 1 grm. of the finely-pulverized slag into a 3½-in. casserole, moisten with a little water to prevent sticking, cover with a clock glass, and add 20 cc. of hot acid (1 HNO_3, 3 HCl, 4 H_2O). Evaporate slowly to dryness on a hot plate and bake (without overheating) until all the gelatinous silica is dehydrated. Cool, add 20 cc. HCl (1–1), and boil gently for several minutes; filter into a 300-cc. beaker, cleaning out the casserole thoroughly with a rubber-tipped glass rod. Wash residue with hot distilled water and transfer it to a No. 1 porcelain crucible. Dry, ignite, cool in a desiccator and weigh as *total insoluble matter*.

Dilute filtrate to 200 cc. and neutralize some of the excess HCl with NH_4OH. Warm, and pass H_2S for 15 min. Filter and wash with hot water. Boil off H_2S, oxidize with 5 cc. HNO_3, reduce bulk of the solution by evaporation to about 100 cc., and transfer to a 250 cc flask.

SILICA.

Fuse insoluble residue over a blast lamp in a platinum crucible with three or four times its weight of anhydrous carbonate of soda for 10 to 15 minutes, or until decomposition is complete. Cool, and extract fusion in an 8-in. evaporating dish with HCl (1–1); rinse crucible and cover with distilled water, and evaporate to dryness over a water bath. Take up dry mass with 10 cc. strong HCl, and then

50 cc. of hot water; heat to boiling, and filter into the flask containing the original filtrate. Detach adhering silica with a rubber-tipped glass rod. Wash well with hot water. Make the solution up to the 250-cc. mark, cool, and mix thoroughly. Transfer filter paper with silica to a No. 1 porcelain crucible, dry, ignite, cool in a desiccator, and weigh as SiO_2.

FEO, CAO, MGO.

Pipette 100 cc. of solution from the flask into a 300-cc. cylindrical beaker, add 1 grm. of NH_4Cl, 10 cc. of bromine water, and make alkaline with NH_4OH. Boil for a few minutes, and add another 10 cc. of bromine water and again boil. Filter into a 600-cc. beaker and wash well with hot water. (Reserve filtrate for CaO and MgO.) Dissolve the precipitate out of the filter with hot HCl (1–1), catching the solution in the original beaker, and at the same time dissolving any ferric hydroxide adhering to the sides of the beaker. Heat to boiling, and then reduce with $SnCl_2$ solution, add 20 cc. of saturated $HgCl_2$ solution, and titrate with standard $K_2Cr_2O_7$ solution. (1 cc. = 0.01 grm. FeO.)

$$\text{cc. used} \times \frac{5}{2} \times 0.01 \times 100 = \%\ \text{FeO}.$$

Heat filtrate reserved for CaO and MgO to boiling, add 10 cc. saturated ammonium oxalate solution, and continue boiling for five minutes longer. Allow to stand in a warm place for 15 minutes. Filter and wash the precipitate until the washings give no test for oxalate. (Reserve filtrate for MgO.) To the original beaker add 20 cc. H_2SO_4 (1–1) and 200 cc. boiling water, and to this add the filter paper containing the calcium oxalate precipitate, stir well, then titrate immediately with standard $KMnO_4$ solution. (1 cc. = 0.005 grm. CaO.)

$$\text{cc. used} \times \frac{5}{2} \times 0.005 \times 100 = \%\ \text{CaO}.$$

Pass H_2S into the filtrate reserved for magnesia for 15 minutes, boil off H_2S, and at the same time reduce the bulk to about 75 cc. Filter through a double filter paper and wash with hot water several times. Make the filtrate just acid with HCl, bring to boil, and add a few cc. of a solution of microcosmic salt while boiling, cool, add one-third the bulk of a 10% solution of NH_4OH. Allow to stand 12 hours, stirring at intervals. Filter through a double filter paper and wash with $2\frac{1}{2}$ % ammonia water. Transfer precipitate to a No. 1 porcelain crucible. Dry and ignite, first at a low temperature and finally at a bright red heat, to constant weight. Weigh as $Mg_2P_2O_7$.

$$\text{Weight of precipitate} \times \frac{5}{2} \times 0.3624 \times 100 = \%\ \text{MgO}.$$

MnO.

Pipette 50 cc. of solution from the flask into a 500-cc. cylindrical beaker, add 5 cc. HNO_3, and bring to boil. Remove from heat and add an emulsion of zinc oxide and water until iron is precipitated. Make bulk up to 300 cc. with boiling water, and titrate with standard $KMnO_4$ solution. (1 cc. = 0.0038 grm. MnO.)

Titration times 5 × 0.0038 × 100 = % MnO.

Al_2O_3.

Wash out the remaining 100 cc. of original solution from the flask into a 600-cc. beaker. Add 15 cc. of a saturated solution of microcosmic salt, and then add ammonia cautiously and with constant stirring until a slight permanent precipitate forms. Now add 5 drops concentrated HCl, which should dissolve the precipitate and leave a clear solution, then add with constant stirring 20 cc. of a saturated solution of $Na_2S_2O_3$. Dilute to 300 cc., cover, and heat to boiling. When boiling add 20 cc. of a solution consisting of 100 grm. sodium acetate, 200 cc. acetic acid sp. gr. 1.04, and water to make 500 cc. Boil the solution 10 minutes longer. Let the precipitate settle, filter, and wash well with hot water. Now transfer the precipitate to the original beaker, dissolve in HCl, filter, and re-precipitate as described above. Dry and ignite precipitate, cool in a desiccator, and weigh as $AlPO_4$.

$$\text{Weight of precipitate} \times \frac{5}{2} \times 0.4185 \times 100 = \%\ Al_2O_3.$$

ZINC.

Weigh 1 grm. of slag into a $3\frac{1}{2}$-in. casserole, cover with a clock glass, add 15 cc. strong HNO_3, and heat over a Bunsen flame until red fumes of nitrogen peroxide have all been given off. Add 1 grm. of finely-powdered $KClO_3$, bring to boil over a Bunsen flame, using a rotary motion, and then evaporate slowly on asbestos hot plate almost to dryness. Add 30 cc. strong HNO_3 and a little $KClO_3$, heat to boiling, and allow to stand on hot plate 10 minutes. Filter through asbestos pulp and glass wool, detaching adhering insoluble material with a glass rod. Wash twice with HNO_3 (1–1) and several times with hot water. Add 5 grm. of cream of tartar and make alkaline with ammonia. Heat to 70° C. and titrate with standard $K_4Fe(CN)_6$ solution, using glacial acetic acid as an outside indicator.

$K_4Fe(CN)_6$ solution (1 cc. = 0.01 grm. Zn.)

Alternate Method.—Weigh 1 grm. of slag into a 10-oz. conical beaker, add 5 cc. HCl, and evaporate gently to dryness; continue heating at a low temperature until silica is dehydrated. Cool, add 10 cc. concentrated HNO_3, warm gently, then add 1 grm. powdered $KClO_3$, and evaporate to dryness. Allow to cool, then add 50 cc. hot water and 7 grm. NH_4Cl. Stir with a rubber-tipped glass rod to ensure disintegration of the mass, then add 15 cc. NH_4OH and ½ grm. Na_2O_2, cover, and boil for 2 minutes. Filter into a 600-cc. beaker and wash several times with hot 1 % NH_4Cl solution. Re-dissolve the precipitate with as little HCl as possible into the original beaker, and wash residue in filter paper with hot water until free from ferric chloride. Make solution alkaline with NH_4OH, cool, and add 3 or 4 grm. of Na_2O_2. Boil for 2 minutes and filter into the beaker containing the first filtrate. Wash with hot 1 % solution of NH_4Cl. Make solution just acid with HCl, add 5 grm. granulated lead, and boil for a few minutes until copper is precipitated. Make alkaline with NH_4OH, add 30 cc. of tartrate emulsion, heat to 70° C., and titrate with standard $K_4Fe(CN)_6$ solution (1 cc. = 0.01 grm. Zn), using glacial acetic acid as an indicator.

LEAD.

Weigh 5 grm. of slag into a 4-in. casserole, moisten with water, and decompose with 30–40 cc. of hot acid (1 HNO_3, 3 HCl, 4 H_2O). Evaporate slowly to dryness and bake to dehydrate gelatinous silica. Cool, add 20 cc. strong HCl, and boil for 2 or 3 minutes; dilute with hot water and filter into a 600-cc. beaker, keeping as much as possible of the residue in the casserole. Re-treat residue several times with strong HCl, each time diluting and filtering into original filtrate. Finally, transfer residue on to the filter paper and thoroughly wash with hot water. The filtrate, which is about 300 cc., is nearly neutralized with NH_4OH, leaving just sufficient acid to keep the iron in solution. Warm and pass H_2S for at least 30 minutes. Filter and wash with H_2S water. Dissolve precipitate in dilute HNO_3 (1-1), add 10 cc. strong H_2SO_4, and evaporate to fumes. Cool, dilute with cold water, allow to settle, and filter off $PbSO_4$. Wash twice with cold water, then place filter paper and $PbSO_4$ in the original beaker. add 20 cc. ammonium acetate solution, 150 cc. hot water, boil, and titrate with standard ammonium molybdate solution (1 cc. = 0.01 grm. Pb). As the above method is too long for ordinary routine work, a much quicker method is substituted.

Alternate Method.—Weigh 1 grm. of slag into a 3½-in. casserole, moisten with water, and add 20 cc. hot acid (1 HNO_3, 3 HCl, 4 H_2O). Evaporate to dryness without a clock glass and bake carefully. Add 20 cc. H_2SO_4 and evaporate until only a few cc. are left. Cool. add a little cold water, and dilute with boiling water. Allow to stand until sulphate of iron is dissolved and then filter. Wash twice with hot water. Place precipitate and filter paper in a beaker containing ammonium acetate and hot water, boil, stir well, and titrate with standard ammonium molybdate solution.

SULPHUR.

Weigh 1 grm. of slag into a 3½-in. casserole, cover with a clock glass, and add 20 cc. of a solution of HNO_3 saturated with $KClO_3$. Evaporate slowly to dryness on asbestos hot plate, cool, add 3 grm. anhydrous Na_2CO_3, 20 cc. hot water, and boil for 5 minutes. Filter into a 600-cc. beaker containing 15 cc. HCl. Detach adhering residue from the casserole with a rubber-tipped glass rod. Wash precipitate in filter paper four or five times with boiling water. Heat the filtrate to boiling, add 5 cc. of a saturated solution of $BaCl_2$, and continue boiling for 5 minutes longer. Allow precipitate to settle, filter, and wash with hot water until free from $BaCl_2$. Place precipitate in a No. 1 porcelain crucible, dry, ignite, cool in a desiccator, and weigh as $BaSO_4$. Factor for S = 0.1373.

CHLORINE.

Weigh 5 grm. of slag into a 300-cc. conical beaker, add 5 grm. Na_2CO_3 and 100 cc. hot water. Boil for 10 to 15 minutes. Filter into a 500-cc. beaker containing 20 cc. HNO_3. Wash residue five or six times with hot water. Heat the filtrate to boiling, add a few cc. of $AgNO_3$ solution, and continue heating until the precipitate of AgCl has coagulated. Filter and wash until free from $AgNO_3$. Place filter paper containing precipitate in a small scorifier, dry, cover with a little granulated assay lead and litharge, and scorify. The resulting lead button is then cupelled and the silver prill weighed. Factor for chlorine = 0.3287.

COPPER, ARSENIC, ANTIMONY.

Weigh 10 grm. of slag into a 4-in. casserole, cover with a clock glass, add 50 cc. HNO_3, and heat until thoroughly decomposed. Add 40 cc. H_2SO_4, and evaporate to strong fumes. Cool, dilute, add a few grams of tartaric acid, and boil until all soluble sulphates are dissolved. Allow to settle, filter into an 800-cc. cylindrical beaker, retaining residue in the casserole. Wash by decantation until free from tartaric acid. The residue is now decomposed with 30 to 40 cc. HCl, fumed again with 20 to 30 cc. H_2SO_4, diluted, tartaric acid added, boiled, and filtered as previously. To the filtrate add NH_4OH, until most of the excess acid has been neutralized. Warm on hot plate and pass H_2S for 30 minutes. Allow to stand in a warm place for several hours, or preferably overnight. Warm and pass H_2S again for 20 minutes, allow to settle, filter, and wash well with H_2S water. Open the filter paper, and by means of a jet of hot water wash the precipitate into a 300-cc. beaker; dissolve any precipitate adhering to the filter paper with a little hot dilute HNO_3. Make the solution alkaline with NaOH, and warm to dissolve the sulphides of arsenic and antimony. Dilute with hot water to 100 cc., add 5 cc. Na_2S solution, and allow to stand in a warm place for 15 minutes. Filter and wash with warm dilute Na_2S solution. (Reserve filtrate for arsenic and antimony.) Dissolve the precipitate in dilute HNO_3, filter off sulphur, make alkaline with NH_4OH, cool, and titrate with standard KCN solution (1 cc. = 0.005 grm. Cu).

Make the filtrate reserved for arsenic and antimony slightly acid with HCl, pass H_2S for 30 minutes, allow to stand in a warm place for 30 minutes longer. Filter and wash with H_2S water. Dissolve the precipitate off the filter paper with a little hot dilute NaOH solution, add 1 grm. of powdered $KClO_3$, and make distinctly acid with HCl. Allow solution to digest on hot asbestos plate without boiling until the sulphides of arsenic and antimony are decomposed and the solution is free from chlorine. Filter off sulphur, add 1 grm. tartaric acid. Make just alkaline with NH_4OH, add 10 cc. "magnesia mixture," and one-third bulk of 10 % NH_4OH. Allow to stand 12 hours, stirring well at frequent intervals. Filter on to a weighed Gooch crucible, using the filter pump. Wash with 2½ % NH_4OH until free from chlorides. Dry at 100° C., and ignite in a muffle, first at a dull red and then at a bright red heat. Cool in a desiccator and weigh as $Mg_2As_2O_7$.

Factor for arsenic = 0.4827.

Boil off excess of ammonia from the filtrate from the arsenic, and reduce the bulk to about 100 cc., make acid with HCl, warm, and pass H_2S for 20 minutes. Filter on to a weighed Gooch crucible, wash well with hot water, heat in a current of CO_2 at 300° C. for 1 hour. Cool to 100° C. in CO_2, and finally in a desiccator. Weigh as Sb_2S_3.

Factor for antimony = 0.7146.

SILVER.

The following charge is used :—

1 A.T. slag.
25 grm. Na_2CO_3.
20 grm. borax. F. Crucible.
20 grm. litharge.

Sufficient sulphur is generally present to reduce the lead button ; if not, use a little flour. The fusion takes 25 minutes, and a fairly high temperature is required. A small cupel is used for cupellation of the resulting lead button.

GOLD.

The following charge is used :—

20 A.T. slag.
300 grm. Na_2CO_3.
300 grm. $Na_2B_4O_7$.
50 grm. PbO. L. Crucible.
3 grm. flour.
1 4-in. nail.

A fairly high temperature is required for the fusion. The above charge gives a button which may be cupelled directly in a large cupel. The resulting silver prill is parted with HNO_3 in the usual way.

Notes on the Method.—Blank determinations should be carried out along with the assays. Should the slag be very low in lead, copper, arsenic, and antimony, separation of the Group 2 metals may be dispensed with without affecting the results.

FeO, CaO, MgO.

Bromine water is added to the ammoniacal solution for precipitation of the manganese. As it is essential for all the manganese to be separated at this stage, two lots of bromine are added to ensure this result. Should the manganese not be completely precipitated it will contaminate the $Mg_2P_2O_7$ at a later stage. It is not essential to separate the zinc before precipitating the magnesia, but it is advisable, as some zinc may be precipitated along with it if this is not done.

ZINC.

The end point obtained in the first method for zinc is a greenish colour, due to the large amount of iron in the slag.

In standardizing the solution of $K_4Fe(CN)_6$ for this method, iron wire approximately equivalent to the iron content of the slag is added.

Alternate Method.—In some cases the slag under treatment is not entirely decomposed by the ordinary acids, and the residue requires further treatment with HF and dilute H_2SO_4 for complete liberation of the zinc.

"TARTRATE EMULSION."

450 grm. cream of tartar, 45 cc. of a 10 % solution of neutral ferric chloride, made up to 3 litres with distilled water.

A typical analysis of lead blast-furnace slag as produced at the B.H.A.S. Pty. Ltd., Port Pirie, is as follows :—

	%	
SiO_2	20.1	
FeO	26.8	
MnO	5.3	
CaO	12.4	
MgO	0.8	
Zn	17.6	(equivalent to 22.0 % ZnO)
Al_2O_3	7.0	
S	2.2	
Cu	0.25	
As	0.027	
Sb	0.065	

PROCEEDINGS AUS. I.M.M. (INC.)] [N. S., NO. 47, 1922.

THE DETERMINATION OF DUST LOSSES IN METALLURGICAL PROCESSES, AS PRACTISED AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

By A. J. HEMINGWAY.

IN carrying out experiments to determine the amount of metal taken away by fume or fine dust the apparatus used requires to be strong and compact, as it is generally necessary to work on a staging fixed a considerable distance above the ground and against a smoke stack or flue. It is essential that the test be carried out at a part of the stack well away from any bends, to prevent eddies. The method described below has been used for estimating the dust losses from Dwight & Lloyd sintering machines, antimony drossing furnaces, pot sintering trials, etc., and has been found satisfactory. It is well to bear in mind that the longer the period over which dust collection is extended, the more valuable the result is likely to be. The rate of aspiration of the gases through the apparatus is kept as regular as possible by changing the filter thimbles frequently, and the rate of flow should correspond closely with the rate of flow of the gases up the stack.

The apparatus required for carrying out a test is as follows:—

(a) *Thermometers*:—Two are required, one for taking the temperature of the gas in the stack and one for giving the temperature of the gas in the aspirating drum. If the temperature in the stack is over 400° C., it is advisable to use a pyrometer such as the "Bristol," the couple being standardized before use at about the temperature of the stack.

(b) *Pitot Tube*:—The following sketch (Fig. 1) illustrates the principle of the pitot tube. The readings are to be taken in a straight flue, or stack, preferably 20 diam. away from any bend. A, which is open, faces the air current and gives static and impact pressure = pressure in A to one arm of manometer. B, when connected to the manometer, gives the static pressure. When A and B are connected to the two arms, the manometer records impact pressure.

If the stack is small, say not more than one foot in diameter, it is only necessary to place the pitot tube so that the end is in the centre of the stack, as this will give a fair indication of the velocity of the gases. This can be verified by moving the tube so that the end is in different positions in the stack and noting if there is any appreciable variation in the readings of the manometer. When the flue or stack is a large one it is necessary to make a careful traverse in order to accurately determine the average velocity. If the flue is rectangular, it must be divided into small rectangles, 12 or more, and the velocity determined in the centre of each. If the stack is circular it is divided into three or more equal annular areas and four readings are taken on the central lines of each area (central meaning that each annular area is divided into two equal areas).

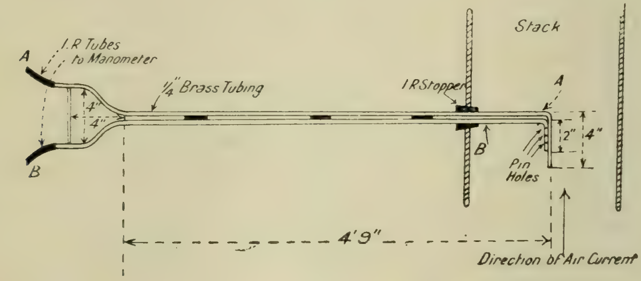


Fig. 1.
Pitot Tube.

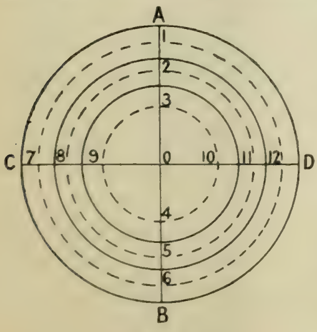


Fig. 2.
Pipe Traverse.

Fig. 2 shows the pipe divided into three equal areas, and the central areas are shown dotted.

The following table gives the distance from the edge of the pipe to the point on the traverse in percentage of pipe diameter when three equal areas are used :—

Point.	Percentage of diam.	Example with Pipe 30 in. diam.
		in.
1	4.5	1 = 1.35
2	14.7	2 = 4.41
3	29.6	3 = 8.88
4	70.4	4 = 21.12
5	85.3	5 = 25.59
6	95.5	6 = 28.65

A traverse of the pipe is first made along A B and readings taken at points 1, 2, 3, 4, 5 and 6, then readings are taken at points 7, 8, 9, 10, 11 and 12.

The velocities are calculated for each point and then averaged. The formula given below is used for calculating the velocity and volume of gas in cub. ft. per minute.

Development of Pitot Tube Formula:—To determine the velocity of gases in pipes.

$V = \sqrt{2gh}$, where V = velocity in ft. per sec.
$g = 32.16$
h = height of air column which would give same pressure as water manometer.

Weight of 1 cub. ft. of water at 0° C. = 62.347 lb.
,, ,, 1 ,, ,, air at 0° C. = 0.08071 lb. and 760 mm.

$$h = \frac{62.347\ H}{12 \times 0.08071}$$ where H = manometer reading in inches of water.

At 0° C. and 760 mm.

$$V = \sqrt{\frac{2 \times 32.16 \times 62.347 \times H}{12 \times 0.08071}}$$

$$= 64.35\ \sqrt{H}$$

At any given temperature (T) and pressure (P)

$$V = \sqrt{\frac{2 \times 32.16 \times 62.347 \times H}{12 \times 0.08071 \times \frac{P}{760} + \frac{273}{273 + T^{\circ} C.}}}$$

The volume in cub. ft. per minute = V × 60 × area of pipe.

(c) *Differential Manometer :*—By using a differential manometer of the type illustrated in Fig. 3, accurate readings are obtained.

This is especially the case when the velocities are low, as the gauge can be made to give a good magnification. The manometer requires kerosene, the density of which is about 0.79, and alcohol, the density of which is made 0.89 by the addition of water, thus giving a difference of 0.1 in density. The alcohol is coloured with a dye such as methyl violet or carmine in order to give a sharp contrast at the line of meeting of the two liquids.

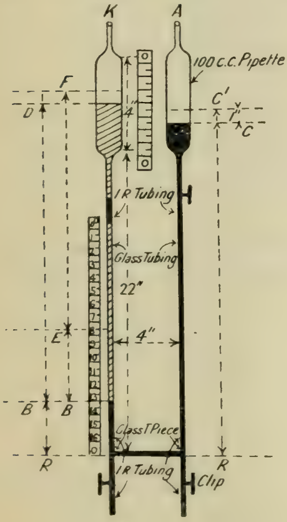


FIG. 3.
DIFFERENTIAL MANOMETER.

Method of Standardizing :—The level is noted in A then the rubber tubing on K is pinched with a clip and 1 in. of alcohol (density 0.89) run into bulb A. The clip is then released, and the rise in level in the tube noted. This represents the rise equal to the pressure of 1 in. of alcohol = X.

Therefore $\frac{X}{0.89}$ = rise due to 1 in. of water = factor of manometer.

The rise, being a function both of the ratio of bulbs to tubes and the difference in gravities of fluids, gauges of varying magnification may be constructed.

The standardizing may also be done by connecting bulb A by means of rubber tubing to a compressed-air pipe and depressing the level of the alcohol in A one inch. The rise in tube K is then noted.

(d) *Bent glass tube :*—$\frac{}{16}$ in. diam. is a convenient size. The length is varied according to the diameter of the stack.

(e) *Filter thimbles:*—Ordinary thimbles such as those used in the Soxhlet extractor.

(f) *Iron drums:*—Two are required for aspirators. They should be about 60 litres capacity and have a $\frac{3}{4}$-in. tap at bottom and glass side tube to act as gauge.

(g) *A length of $\frac{3}{4}$ in. hose.*

(h) *Iron nozzles:*—A series of iron nozzles with holes of the following sizes :—$\frac{1}{8}$ in., $\frac{3}{16}$ in., $\frac{1}{4}$ in., $\frac{5}{16}$ in., $\frac{3}{8}$ in., $\frac{7}{16}$ in. and $\frac{1}{2}$ in.

Fig. 4 shows the method of setting up the apparatus.

A bent glass tube, whose area bears a known ratio to the area of the stack or flue, is placed in position. The volume of gas flowing per minute at the temperature of the stack is obtained by means of the pitot tube, differential manometer and thermometer. The equivalent amount of gas in litres per minute, which at the temperature

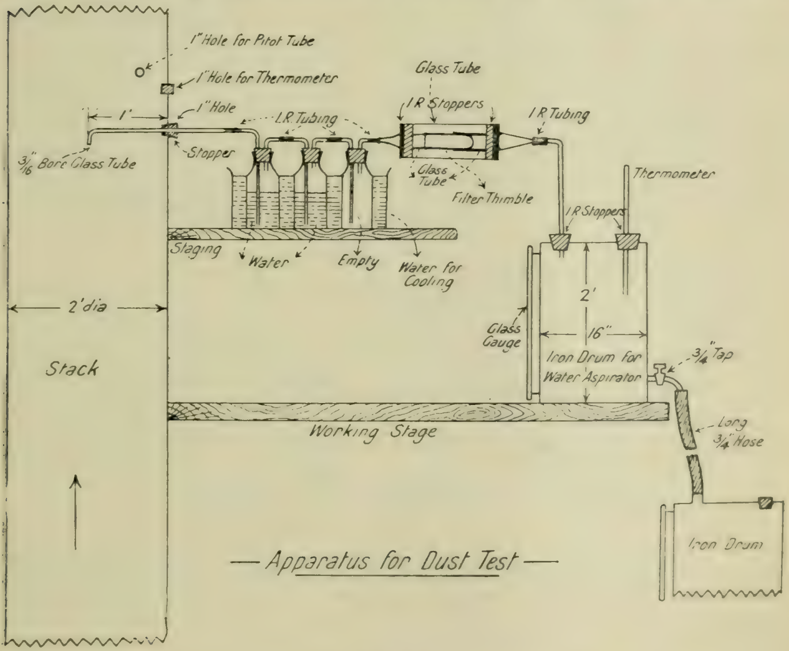


FIG. 4.

of aspiration has to flow through the glass tube, is then deduced. To secure this velocity, aspiration by means of a strong iron cylindrical drum of about 60 litres capacity with glass tube gauge is used.

The gauge is graduated in litres. The flow of water from the aspirator may be regulated by using one of the series of iron nozzles and by altering the length of the hose. The tap on the drum also helps in regulating the flow. The following example illustrates the method of calculating the rate of flow in litres per minute :—

Diameter of glass tube in stack = $\frac{3}{16}$ in.
Diameter of stack = 2 ft.
Area of cross section of stack = 3.1416 sq. ft.
Ratio of area of glass tube to area of stack $= \dfrac{\left(\dfrac{3}{32 \times 12}\right)^2}{1}$

$$= \left(\frac{3}{32 \times 12}\right)^2$$

Suppose the volume of gas was found to be 6139 cub. ft per minute at 110° C. and the temperature of aspiration was 20° C.

$$\text{Volume of gas} = \frac{6139}{1} \times \frac{273 + 20}{273 + 110} \text{ cub. ft. per minute at } 20^\circ \text{ C.}$$

(1 litre = 0.0352 cub. ft).

Therefore, litres to be aspirated =

$$\frac{6139}{1} \times \frac{273 + 20}{273 + 110} \times \frac{1}{0.0352} \times \left(\frac{3}{32 \times 12}\right)^2 \text{ litres per minute.}$$

= 8.14 litres per minute.

Having selected a suitable nozzle. the drum is filled and the time for running out 60 litres of water should be $\frac{60}{8.14}$ minutes.

Marks are also made on the glass gauge of the aspirator drum to show one minute divisions so that the speed of the flow can be kept fairly uniform. This rate is obtained by means of a hose (attached to the $\frac{3}{4}$-in. tap of the drum) which is made to act as a syphon, and to the end of the hose is fixed the nozzle with a small hole, say $\frac{1}{4}$ in. diam. This nozzle is plugged with a small wooden peg when the syphon is not running.

METHOD OF OPERATING.

The hose being a complete syphon, the small plug on the nozzle is removed and the time taken for aspirating 60 litres. This should be $\frac{60}{8.14}$ minutes, but will perhaps vary slightly from this at times. When the filter thimble becomes choked the time of aspiration will begin to increase and a new thimble (previously dried at 100° C. weighed and kept in a desiccator) must be inserted. This aspiration is practically continuous if two drums are used (operating and being filled alternately). A thermometer in the drum records the temperature of collection. The pitot tube with manometer and thermometer should be in position all through the experiment and readings taken at close intervals. In the sketch of the apparatus used in the determination (Fig. 4) three bottles are shown, two of which contain water. The bottles are kept cool by placing them in a trough containing water, which is replaced at intervals. This prevents the water in the bottles from becoming too hot. These bottles are only used to prevent the thimbles choking too quickly when a large quantity of dust is passing through the stack. When only a small quantity of dust is passing, one empty bottle should be sufficient.

TREATMENT OF THE CATCH.

The contents of the bottles and connecting tubes are placed in a weighed porcelain dish (if the weight of catchment is required), the tubes being cleaned out with water. The water is evaporated off and the weight of residue obtained. The bent glass tube from the stack is weighed before and after the trial, care being taken to remove any dust from the outside of this tube. The filter thimbles are weighed before and after the trial. They are dried at 100° C in an air oven and kept in a desiccator when not in use. The total weight of dust caught is thus obtained. If only the amounts of lead, zinc, etc., in the dust are required the above evaporation may not be necessary, as these could be estimated.

CALCULATION.

Weight of dust = X grm.

Total time of aspiration = Y minutes.

$$\text{Ratio :—}\ \frac{\text{area of stack}}{\text{area of tube}} = \left(\frac{32 \times 12}{3}\right)^2$$

$$\text{Tons of dust per hour} = \frac{X}{453.6} \times \frac{1}{2240} \times \frac{1}{Y} \times \frac{60}{1} \times \left(\frac{32 \times 12}{3}\right)^{2}$$

Should volume of gas aspirated be slightly greater or less per minute than was computed, a correction—

$$\frac{\text{amount which should have been aspirated per minute}}{\text{amount actually aspirated per minute}}$$

may be applied, but this ratio should be very close to unity.

THE ANALYSIS OF REFINED LEAD, AS PRACTISED AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

BY A. J. HEMINGWAY.

IN preparing a sample of refined lead for analysis, it is necessary to exercise great care to avoid contamination, as the amounts of the various impurities present in market lead are very small.

The lead is cut into convenient-sized pieces, which are flattened out with a hammer, so that, on putting the lead through the rolls, strips of about one inch in width and 1/32 in. in thickness are obtained. The hammer, anvil, and rolls should be carefully cleaned before use. The strips of lead are cut up into pieces of about one inch square. If the lead is rolled out very thin and cut up into small pieces there is considerable risk of the acid acting too violently and causing loss through frothing over. A blank assay on all the reagents used should be carried right through, along with the actual analysis of the lead. This is of the greatest importance, particularly with regard to the determination of such impurities as iron, sulphur, arsenic, and antimony.

METHOD OF ANALYSIS.

The method of analysis is as follows:—Take two lots of 600 grm each, place in a large beaker, add 2000 cc. distilled water and 750 cc. concentrated HNO_3. Cover with a clock glass and warm on a hot asbestos plate until all action ceases. (If any white precipitate forms, filter off and examine for Sn and Sb.) Add 420 cc. H_2SO_4 (1–1), and stir well while adding the acid. Allow $PbSO_4$ to settle, pour off the clear liquid into a 12-in. porcelain evaporating dish, and start evaporating down, the dish being placed directly on an 8-in. ring gas burner. Wash the precipitate three times by decantation, using approximately 1200 cc. hot water for each washing, and stir well each time. Filter through filter paper on a 6-in. glass funnel, then pass the whole of the precipitate on to the filter paper, wash three times, and allow to drain. Gradually add the washings to the main bulk of solution in the dish. Evaporate right down to fumes, leaving not more than 10 cc. of solution. Cool, add 50 cc. cold water, heat gently to boiling for 10 min., and then add 2 grm. tartaric acid. Wash down the sides of the dish with a jet of hot water, then filter and wash three times. Re-treat the $PbSO_4$ by washing it out of the filter paper into 300-cc. beakers, pour off the liquid through the same filter, then boil up with 15 cc. strong HCl and a small amount of $KClO_3$ until the $PbSO_4$ is broken up. Add 10 cc. concentrated H_2SO_4, fume, cool, dilute, add 1 grm. of tartaric acid, boil, and filter into the first filtrate. Re-treat the $PbSO_4$ in the same way a second time, and filter into the beaker containing the last two filtrates. The bulk of the solution is now about 400 cc. Add 10 cc. strong HCl, warm on the asbestos plate, and pass a rapid current of H_2S for 45 min. Allow to stand in a warm place overnight. In the morning again pass H_2S for 15 min., then filter and wash with H_2S water. The filtrate contains Zn, Cd, Ni, Fe, and traces of Pb (filtrate 1). Precipitate contains Cu, Bi, As, Sb, and Pb (precipitate 1). Wash precipitate back into the 400-cc. cylindrical beaker containing a few crystals of tartaric acid and dissolve any precipitate adhering to the filter paper with hot HNO_3 (1–1). Add NaOH until alkaline, and then add 15 cc. Na_2S solution (250 grm. dissolved in 500 cc. water). Make bulk up to 200 cc., allow to stand in a warm place for 10 min., filter and wash with warm water containing a little Na_2S.

Filtrate from Na_2S Treatment.—Make acid with HCl, add 10 cc. in excess, pass H_2S into the hot solution for 10 min. and allow to settle for two hours. Filter and wash twice with H_2S water. Wash the precipitate back into the same beaker and dissolve any adhering to the filter paper with a little hot dilute NaOH solution. Add 20 cc. of HCl (if the bulk is kept low enough, this amount of acid will be sufficient), 1 grm. of $KClO_3$, and heat carefully until the Sb and As are in solution, and some sulphur separates. Add 0.5 grm. tartaric acid, filter and wash with hot water several times, keeping the bulk of the solution as low as possible. Arsenic—Cool, make alkaline with NH_4OH, add 20 cc. "magnesia mixture," then one-tenth of its bulk of NH_4OH, allow to stand in a cool place for two hours, stirring well at frequent intervals. Allow to stand for one hour more, and then filter through a small weighed Gooch crucible containing an asbestos wad. Wash with NH_4OH water, dry in hot-air oven, then heat in a muffle at a red heat for 15 min. Cool and weigh as $Mg_2As_2O_7$ ($\times$ 0.4838 = As).

Filtrate from Mg_2As_2O .—Boil until most of the NH_4OH is expelled. Make acid with concentrated HCl and add 10 cc. in excess. Boil for 20 min. to expel any chlorine. Antimony—Pass H_2S through the warm solution for 30 min., allow to settle in a warm place, and filter on to a weighed Gooch crucible. Wash several times with warm water. Heat in an atmosphere of CO_2, raising the temperature gradually, and keep for 20 min. at a temperature of 270° C. Cool to 100° C. in CO_2, and finally in a desiccator. Weigh as Sb_2S_3. ($\times$ 0.7146 = Sb.)

Precipitate from Na_2S *Treatment.*—Dissolve off the filter paper with warm HNO_3 (1–1), then add 7 cc. concentrated H_2SO_4, and take to strong fumes. Cool, dilute to 100 cc., filter, and wash with cold water containing a little HS_2O_4. This separates the last traces of Pb. Bismuth—Pass H_2S for 15 min. into the hot filtrate, allow the precipitate to settle, filter, and wash with H_2S water. Re-dissolve the precipitate back into the beaker with warm HNO_3 (1–1), heat until all is decomposed, then add 2 drops of HCl and stand in a warm place until the AgCl settles out. Filter, make filtrate alkaline with NH_4OH, and boil. Add 15 cc. $(NH_4)_2CO_3$ solution (10 %), bring to the boil again, and allow to settle for 20 min. Filter through a weighed Gooch crucible, wash with hot water, dry, heat in a muffle to dull redness for 5 min., cool, and weigh as Bi_2O_3. ($\times$ 0.8965 = Bi.)

Filtrate from the Bismuth.—Make acid with HCl, boil, pass H_2S to precipitate the Cu, allow precipitate to settle, then filter and wash with H_2S water. Copper—Dry the precipitate and ignite it carefully over a Bunsen burner in a small porcelain crucible. Add 2 cc. strong HNO_3, evaporate down to 0.5 cc., dilute with a little cold water, and add NH_4OH until alkaline. Filter into a larger white porcelain crucible (No. 2) through a small filter paper. Add to the blank assay, containing the same amount of reagents, a standard solution of copper nitrate until the two blue shades are matched. (1 cc. $Cu(NO_3)_2$ = 0.0005 grm. Cu.)

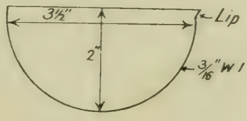
Filtrate 1, *containing Cd, Fe, Zn, Ni.*—Make both assay solutions and the blank assay alkaline with NH_4OH and pass H_2S for 15 min., allow to settle for 10 min., then filter, but do not wash. Re-dissolve the precipitate off the filter paper with 10 cc. HNO_3 (1–1), boil until all is decomposed, add 6 cc. H_2SO_4, take to fumes, cool, dilute and filter, washing with cold water. This removes the last traces of Pb. Cadmium—Make bulk up to 200 cc., heat to boiling, and pass a rapid current of H_2S for 20 min. to precipitate Cd. Filter and wash. If much Cd is present, re-dissolve the precipitate in 10 cc. warm HCl (1–1), take to fumes with 6 cc. H_2SO_4, cool, make bulk up to 200 cc., and re-precipitate the CdS. Dissolve the precipitate off the filter paper with hot HCl (1–1) into a weighed platinum dish, add 5 drops of H_2SO_4, take to dryness, and heat carefully over a small Bunsen flame until all fumes cease. Cool in a dessicator, and weigh at $CdSO_4$. ($\times$ 0.539 = Cd.)

Filtrates from Cd.—Add NH_4OH until alkaline and pass H_2S for 10 min. Allow to settle, and then filter, but do not wash. Re-dissolve the precipitate off the filter with hot HNO_3 (1–1), and warm until all is decomposed. Iron—Add 1 grm. of NH_4Cl, and then NH_4OH until alkaline, boil, settle, and filter. Re-dissolve the precipitate with HCl, re-precipitate with NH_4OH, and filter into the first filtrate. Dissolve the precipitate with HCl, reduce with $SnCl_2$, add $HgCl_2$ in excess, and titrate with standard $K_2Cr_2O_7$ solution. (1 cc. = 0.00475 grm. Fe.)

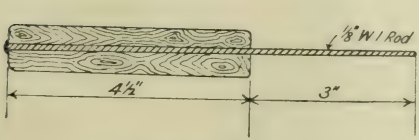
The filtrate from the Fe contains Zn and Ni, if present in the sample. Pass H_2S, and while the H_2S is passing add 6 drops of ferric chloride solution (10 %). Allow the precipitate to settle, filter, re-dissolve the precipitate with 10 cc. HNO_3 (1–1), and boil until all the precipitate is decomposed. Zinc—Now continue with both assay and blank by adding 5 grm. of cream of tartar, 2 grm. NH_4Cl, and make alkaline with NH_4OH, then add 0.5 cc. in excess in a bulk of 150 cc. Titrate at 70° C. with standard dilute $K_4FeC_6N_6$ solution (1 cc. = 0.002 grm. Zn), using glacial acetic acid as outside indicator.

Note.—The blank assay is started off by placing 750 cc. concentrated HNO_3 and 210 cc. concentrated H_2SO_4 in the 12-in. porcelain basin and evaporating it down along with the assays. It is then taken right through all the estimations given above, in the same manner as the assays.

Sulphur.—Weigh out two lots of 500 grm. each of the rolled lead, cut into small pieces. Heat three wrought-iron pots over a Bunsen flame, fitting each pot into a hole in a piece of asbestos board. Place in each pot 40 grm. of NaOH. (Take a fresh bottle of NaOH; break the NaOH up quickly into small pieces in a porcelain mortar and mix it up well.) When the NaOH is molten, add the lead piece by piece until all is melted, stirring well all the time. In one of the pots run a blank charge, using some lead which has been treated several times with NaOH as described above. Keep molten for 15 min., stirring well all the time. Cool, and extract with hot water, decanting into a 600-cc. beaker. Add 20 cc. bromine water and allow to stand for a quarter of an hour on a warm asbestos plate, then make acid with HCl and boil off Br. Filter and wash with warm water. Heat filtrate to boiling and add $BaCl_2$ solution. Boil, and then allow to settle in a warm place overnight. Filter, wash the precipitate well with hot water. Dry, ignite, and weigh as $BaSO_4$. ($\times$ 0.1373 = S.)



—W.I. Cup—



—Stirrer—

Fig. 1.

Silver.—Triplicate assays of 3 A.T. each are cupelled directly in large cupels.

Gold.—If the lead is to be tried for gold, or, in the case of a lead very low in silver, duplicate assays of 36 A.T. each are scorified, using 6 scorifiers (3½-in.) for each assay. The six lead buttons, when sufficiently small, being combined, cupelled together, and the silver prills weighed. The two prills are then parted with HNO_3, and any gold determined. The amount of gold present in refined lead is generally so small that it is often necessary to take larger quantities of the sample for the estimation.

Note.—When nickel is present, proceed in the following manner :—After precipitating the iron, make the filtrate slightly acid with HCl, then add NH_4Cl and enough NH_4OH to make it slightly ammoniacal. Nickel—No precipitate will be formed if sufficient NH_4Cl has been added. The solution is then just acidified with HCl, heated to boiling, and the nickel precipitated with an alcoholic 1 % dimethylglyoxime solution, boiled for a few minutes, and allowed to settle. While still hot the precipitate is filtered into a weighed Gooch crucible, washed with warm water, and dried at 110° C. for 45 min. (20.31 % of the precipitate is Ni.) The filtrate is made slightly acid with acetic acid and H_2S passed to saturation. The precipitate is filtered off, washed with H_2S water, then dissolved in 10 cc. HNO_3 (1–1), and the solution is ready for the determination of Zn.

Manganese is not likely to be present in a refined lead.

TYPICAL ANALYSIS OF A MARKET LEAD.

%
Cu = 0.00027
Bi = 0.00107
Cd = 0.00046
Sn = nil
Sb = 0.00713
As = trace
Fe = 0.00041
Zn = 0.00072
S = 0.00089
Ni = trace
Pb = 99.98850 (by difference)
Ag = 0.00055 = 0.18 oz. per ton

100.00000

Au = 0.025 gr. per ton.

INDUSTRIAL ASSAYING AND A SYSTEM OF COSTING AS PRACTISED AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

BY A. J. HEMINGWAY.

IN any assay office it is of paramount importance to keep the men interested in their work ; and in order to do this, men on routine work, such as lead, zinc, and sulphur determinations, slag analysis, etc., should be changed around monthly. This gives some variety and helps to keep a man's interest on his work, so that better work results. It may be maintained that to keep a man on the same class of work indefinitely makes him more expert at his work. This is certainly true ; but in assaying, where accuracy is so essential, it is absolutely necessary for a man to keep his mind on the work, and not get into a mechanical method of working. He is also on the alert to discover means for shortening his analytical methods, and his capacity for work is increased. Another advantage is that the men get a good all-round training, and one is never at a loss to replace a man on any class of work when he is absent from any cause. This tends to smoother and more efficient running of the assay department. In industrial assaying it is often advantageous to have a result quickly, so that it will be of maximum benefit in the running of some part of the plant. In such a case it pays to sacrifice a little accuracy to speed up a method. The shorter method is less costly, and a quick approximate method is all that is required to give the superintendent the requisite information. Time is thereby saved, and the cost is less. In carrying out assaying for any large industrial plant, such as a smelting works, there is always a tendency for superintendents running the different departments to send in more samples for assay than are absolutely necessary when no system of costing is in operation. Some of these samples may involve a lot of work and heavy expenditure, as, for example, the estimation of only one impurity in a sample of refined lead. If the superintendent knows that the cost of assaying all samples sent in by him is charged up directly against his department, he will consider, before sending in any extra samples, whether the benefit he is likely to derive from having them assayed will warrant the extra expenditure. Each superintendent is supplied with a list showing the cost of every determination likely to be done on samples sent in from his department. By reference to this list he can quickly see what the assaying of any sample is likely to cost his department. It is also far more satisfactory for a chemist or assayer to know that the work he is doing is going to be of considerable help in running the plant, as this acts as an incentive for him to put forward his best efforts. The institution of a costing system in connection with the assay office of any works also tends to decrease the expenditure on materials. The chemist becomes acquainted with the cost of carrying out the various estimations, and he knows that it is in his own interest also to make every effort to keep the cost as low as possible. He learns something about the cost of the materials he uses, and that acts as a check against using too much of any reagent, especially when he knows that a certain reagent is expensive. The head of the assay department naturally watches his cost sheets closely. He is often enabled to cut down expenses and keep a check upon the number of samples sent in for assay. Without a costing system some items of expenditure creep in and are liable to escape notice.

The method adopted for working out the costs of assaying in use at the assay department of the Broken Hill Associated Smelters is, briefly, as follows :—The department is divided into two main parts —namely, chemical laboratories and fire room—the laboratories being further subdivided into No. 1, 2, 3, rooms and silver room, for the purpose of allocating the wages, general charges, and cost of supervision. The average salary of the chemists or assayers working in any given part is taken as a basis for working out the cost, as far as salary is concerned, of any determinations carried out in that part of the office. The cost in wages per eight hours per man is then calculated from the actual time worked. As regards the wages of the clerical staff, labourers, and boys, suppose there are 15 chemists and six assayers employed in the department, two-sevenths of the total salaries of the clerks is charged up to the fire room and five-sevenths to the chemical laboratories. The amount to be added for clerical work to the cost of work done by each assayer or chemist is then worked out in terms of eight hours. If there are two labourers, the wages of one is charged up to the fire room and the wages of the other to the chemical laboratories. Suppose there are four boys employed, the wages of three of these are charged against the fire room, as there is one boy on each shift in the fire room. The wages of the other boy is charged up to the chemical laboratories. The cost of supervision is allocated in the same way as the clerical work. From experience, the average number of assays for lead by the wet method, for example, in an ordinary lead ore that one man could do in a day of eight hours was obtained. The cost of the materials actually used in carrying out these estimations is then computed. The average amounts charged up to the assay department for such items as overhead charges, repairs and renewals to buildings and plant, general stores (not allowed for in working out the cost of materials used in doing any given assay), apparatus, power, lighting, gas, water, coal, coke, etc., for the previous working period of six months are obtained from the costs department. These are all included under one heading—namely, General Charges—and the average amount charged against the assay office for one day of eight hours is worked out. Two-sevenths of this amount is allocated to the fire room and five-sevenths to the chemical laboratories. The total cost of the average number of lead assays which one chemist working in any one of the laboratories could do in a period of eight hours having been worked out, the amount is then divided by the number of assays done, in order to obtain the cost of one lead estimation. The costs of other estimations are derived in a similar manner. The amounts charged for the various determinations are adjusted every 6 months to counteract any variation in wages, etc. Also, by making slight alterations from time to time in the various charges for assays, by either increasing or decreasing the charges according as the amount charged up for assaying is below or in excess of the total expenses involved in the running of the assay department for any monthly costing period, it was found that the amounts charged up could be kept fairly close to the actual expenses. In any case, the works costing department can easily make the necessary adjustment between the different departments. The main object of the assay costing system is to give the different superintendents a good idea of what the assaying is costing their departments. This undoubtedly tends to cut out samples which are not absolutely necessary, and leads to a considerable saving. In addition to that, the sphere of usefulness of the assay office is widened, as time previously taken up in doing plant samples, which were sometimes only used as records of doubtful interest, is available for exprimental work.

It is necessary on a smelting plant to have analyses made, at regular intervals, of the various ores treated, the fluxes and fuel used, the main products, and the slag going to the dump. The results of these analyses are not only of use for entering up as records, but are valuable to the superintendents in making their calculations. Complete analyses are done at the end of each six-monthly period, and the results are of more value than those obtained by doing isolated samples at irregular intervals. The various ores, for example, are sampled, according to the nature of the material, in certain-sized lots as they are received daily, and at the end of each period composite samples are made up according to the tonnage of the particular ore received. The products made are dealt with in a similar manner. Ths method works well when the main supplies of ore, fluxes, and fuel come from known sources, and it results in a saving in the number of samples sent in for analysis. The same principle of preparing composite samples is applied wherever possible in making up input and output samples of material in connection with the various plants. These composite samples from the different departments generally cover a weekly period, but in some cases they may be made to extend over a fortnight, or even a month. This method is quite satisfactory, as the samples give an accurate idea of the materials treated and the products turned out. Calculations based on such analyses are also more reliable. It is far more satisfactory to assay an accurately prepared sample covering, say, a weekly period than to do daily samples, which make considerably more work, and, in addition, increase the possibility of error when the average figures are taken out for statistical purposes at the end of each period.

ACCOUNTING AND COSTING AT THE BROKEN HILL ASSOCIATED SMELTERS PROPRIETARY LTD. WORKS, PORT PIRIE.

BY A. COTTELL, F.C.A.A.

INTRODUCTORY.

As this paper is being written primarily for perusal by mining men, metallurgists, and engineers, an attempt will be made to avoid the use of accounting technicalities. Its scope will be confined to a description of the methods employed in the collection of basic information for use in timekeeping, costing, and the compilation of statistics.

TIMEKEEPING.

Every applicant for work is asked to furnish the following information, which is collected on a card (size 6 in. x 4 in.) :—

Name in full.............................No...........
Date................ Occupation........................
Married.
Year of birth........... Nationality...............Single.
Widower.
Dependents ....
Present address......................................
Union..
Where employed last..................How long.........
In what capacity.....................................
Signature ....Witness................
Date started................as.........................
Department...
Date left or discharged..............................
No. transferred to...................................

After being filled in, this card is filed in the "Applications" drawer, where it remains until the man is employed, when it is transferred to another cabinet, where it is kept in alphabetical order. At the same time another card is inserted in a cabinet in which the cards are filed in numerical order. As soon as the man is engaged he is given a number, which he retains so long as he is employed on the works. For instance, if John Brown was given No. 151, then he would be shown as No. 151 on the pay-roll, on his pay-ticket, on his job ticket, and, in fact, through all the records.

Men are engaged only through the time office, and direct application for work to the head of a department or to a foreman is prohibited. Foremen requiring additional men send in a requisition to the chief timekeeper on the following form :—

B.H.A.S. REQUISITION FOR LABOUR.

Number of men required.................................
Class of labour...
Date required...
Remarks ....
Foreman...
Date...............

The form is in duplicate, one copy being retained by the foreman and the other kept and filed in the time office.

After having selected the men from those available, the timekeepers fill up and send to the foreman the following form :—

ENGAGEMENT ADVICE.

No.........

Form P 11,
B. H. A. S.

To Foreman...........................

The following men have been engaged in response to your requisition dated...................

....
Date. Chief Timekeeper.

No.	Name.	Engaged as.	Rate.	To start date.	Remarks.

If any of these men are unsuitable, or are started on an occupation different from that shown above, the time office must be advised immediately.

This form is in duplicate, one copy being retained by the time office. It will be seen that the foreman is directed to inform the time office should any of the men be unsuitable for the work for which they are required. Should this be the case, the fact is noted on the man's record sheet, and he is not again engaged for that type of work.

Every workman is required to stamp his starting and finishing time on a clock card. This time is recorded by means of the "International" time clocks, which have been used for some 2½ years. It is found that the men can pass these clocks and record their time at the average rate of 30 men per min. The clocks are situated at the main entrance to the works, close to the time office. No trouble has been experienced through wilful attempts on the part of the men to falsify the record or to stamp a card for a fellow workman, but, as a precautionary measure, one of the watchmen is detailed to be present in the clock house when the men are using the clocks.

Any system of timekeeping which relies for its accuracy upon one record only, undoubtedly lends itself to fraud, and a secondary check upon the amount shown on the pay-rolls has been installed. This check, however, has been so devised that it acts as a medium for the collection of information for costing purposes.

Labour at Port Pirie may be subdivided into two main groups—viz., day workers and shift workers. As stated above, the clock house is situated at the main entrance to the works, from which the workshops (where the bulk of the day workers are engaged) are some distance away. It is obvious that, if no precautions were adopted, men could "clock on" at 8 o'clock, but not be on their jobs until 8.30. To guard against this, all day workers are required to attend at the workshops cost office at or before 8 a.m. to receive a starting job-ticket, which is handed to them by one of the cost clerks. This ticket bears the employé's name and number, the date, the starting time, his rate, etc., and a fresh ticket is issued at intervals during the day, whenever the man changes from one job to another. As these tickets are received they are entered on to a "Daily Labour Report." At the end of the day these daily labour reports show the total time worked by each man, and are sent to the time office, where they are compared with the time booked on the clock cards. This comparison is made each day.

So far as the shift workers are concerned, the secondary check is compiled by the various shift bosses, who are required to keep what is known locally as a "Shift Bosses Allocation of Time" record. The time has to be kept under suitable headings for costing purposes, but this gives the shift bosses very little extra trouble. Shift bosses are required to leave their sheets at the time office at the completion of the shift for comparison with the clock cards.

Each day the time-office staff make up a statement showing the total hours worked and the total amount of money payable. This is done as a check against the rates shown on the various records; but, as totals only are used, the amount of work involved is not very great.

After checking the clock cards against the secondary records, the time-office staff enters the time worked on the clock card, at the same time showing thereon the rate to which the man is entitled. The cards are thus available as a means of compiling the pay-rolls. At the end of the fortnight the amount due to each man is worked up and shown on the clock card, and is subsequently transferred to the pay-roll. This latter shows only the man's number, name, gross earnings, deductions, and net amount due. It is obvious that, at the end of the fortnight, the total of the pay-rolls should agree (within very narrow limits) with the daily totals compiled as described above. An additional check on the accuarcy of the additions is provided by the paymaster's abstract of cash required.

It is recognized that nothing tends more to irritation and dissatisfaction amongst workmen than mistakes in their pay, and every effort is made to guard against this happening. So that workmen may have no reasonable ground for complaint, they are furnished with a pay ticket one day before they are paid, which contains the following details :—

THE B. H. A. S. PTY. LTD., PORT PIRIE.

Fortnight ending.................................
No................ Dept.......................
Name ....
Total hrs. (ordinary) ....@....£ s. d.
,, ,, (O.T.)........@.........£ s. d.
Dirt money and allowances....@....£ s. d.
Gross total.
Deductions :—
Advances
Accident Fund
Provident Fund

Net payable .. .. ..

Received the above amount—
......................................Signature.

The men surrender this ticket in return for their pay envelopes.

Workmen are paid in their own time, but every effort is made to expedite payment, and no man has to wait more than 15 minutes for his pay. The pay fortnight ends on Wednesday at 8 a.m., and wages are paid on the following Friday, starting at 8 a.m.

Absences, whether with or without leave, have to be notified by foremen and shift bosses to the time office, and small forms are provided for that purpose. Before resuming work the employé must obtain a "resume note" from the head of his department, which he must subsequently present to the time office.

It is a fixed rule that no officer engaged on the compilation of the pay-rolls shall be allowed to assist in the payment of wages.

Wages unclaimed after the Tuesday following the pay must be paid into the bank to the credit of "Unclaimed Wages Account." The tickets lying unclaimed at the time office are listed in a special book kept for the purpose, the man's number, name, and amount due being shown. The amount is then agreed with the unclaimed money remaining in the paymaster's possession. Unclaimed wages are only paid on presentation of an order, obtained by the employé from the time office, signed by a timekeeper, and countersigned by the chief timekeeper.

As an additional safeguard against fraud, the "Shift Bosses' Allocation of Time Books" have to be presented daily to the head of the department concerned, who initials each day's entries. As will be shown later, these books are subsequently used for costing purposes.

Every workman attending for a specified number of days during the year is entitled to a certain number of days (or shifts) leave on full pay. The leave and absentee notes are entered up daily on the clock cards, and at the end of the fortnight the record is transferred to an "Employee's Bonus Record Book," ruled to provide for the collection of the following information:—

- Hours worked, and rate.
- Non-penalty absences—
    - At conferences.
    - Booked off but attended at gate.
    - Sick, but medical certificate produced.
    - Hurt, but medical certificate produced.
    - Holiday leave.
- Penalty absences—
    - Absent without leave.
    - On leave.
    - Sick—no medical certificate produced.
    - Hurt—no medical certificate produced.
- Time accounted for.
- Remarks.

And on the back of the sheet provision is made for the following:—

SUMMARY.

	hrs.	hrs.
Total hours worked		
Add non-penalty absences—		
At conferences		
Booked off, but attended at gate		
Sick, with medical certificate ..		
Hurt, with medical certificate ..		
Holiday leave		
Other allowed absences		
Total hours credited (fwd)		

	hrs.	hrs.
Total hours credited (fwd)		
Less penalty absences—		
Absent without leave		
Add 50 %		
Absent with leave..		
Sick, no medical certificate ..		
Hurt, no medical certificate ..		
Other penalty absences		
Net credit for bonus		

Voucher date ....Voucher No.........
Bonus due................days @....p. d. £ : :
Signature of certifying clerk.....................
Leave cleared from...........to...........(incl.)

At the end of the year lists are compiled from the above record and sent to the various heads of departments, copies being posted round the works. Men may only clear their leave by permission of the head of the department in which they are working, but every effort is made to meet the convenience of the men. On presentation of a note from the head of the department authorizing the "bonus leave," the employé is given an order on the paymaster for the amount due. This order must be signed by a member of the timekeeping staff and countersigned by the chief timekeeper.

Costing.

Some few years ago, when the maintenance and construction costs as reported came up for criticism, it was found that much of the information was unreliable, owing to the fact that the time worked and the order numbers were recorded by the men themselves. Furthermore, these costs were only reported at periodic intervals, corresponding with the operating costing periods. It was found that the work could not be finalized and the costs reported until perhaps 12 or 14 days after the completion of the job. Frequently the accuracy of the figures was challenged, and although the cost office, in the great majority of cases, was able to prove the accuracy of their figures on the information as furnished to them, it was felt that the information given did not reflect the facts. Moreover, the work had been finished for some time, and, though the job might have cost an unjustifiable amount, no remedy was forthcoming, and general dissatisfaction resulted. It was felt that a costing system which could not produce fresh and reliable data was worse than useless.

Many conferences were held to endeavour to discover where the fault lay, and it was finally concluded that it was useless to rely on the men to furnish accurate records either as to time spent on the job or the job number to which the work was chargeable. Moreover, it was found that the foremen were forced to neglect their proper work of supervision in their endeavours to discover errors made by the men in charging up their time. Finally, it was decided to try a drastic change by establishing a costing office in the shops themselves, together with the system described below.

MAINTENANCE AND CONSTRUCTION COSTING.

At starting time each morning all men on maintenance and construction work are required to present themselves at the workshops cost office to receive a starting job ticket. At the time it is handed to the workman this ticket shows only the date, the time of starting, the man's name, number, and rate of pay. The name of the gang to which the man is attached is printed in bold type on the ticket to facilitate sorting. The man takes the ticket away to the job, retaining it until the job is completed. He then presents it to his foreman, who writes on the ticket a short description of the work the man has been engaged upon, and hands it back to the man, who takes it to the cost office, surrenders it to the clerk at the window, and receives another ticket bearing the time of starting the new job. This process is repeated throughout the day as often as the man changes his job. Exceptions to this procedure are authorized where the men may be working at some distance from the cost office. In these cases the foreman will place the time on the job ticket and hand the man a fresh ticket. The foreman keeps the old ticket until the end of the day, or he may hand it in when passing the cost office. The job number is placed on the job ticket by the cost clerk, who is able to ascertain the number of the job from the description given.

As will be seen from the foregoing, all the foreman is asked to do is to give a short description of the work. He is not held responsible for job numbers, and he can do what is required of him without leaving his men. It might be thought that the necessity for the men to go to the cost office window to change their job tickets would lead to much lost time. The fear of this happening was one of the strongest objections raised before the introduction of the present system, and the matter was very closely watched by the engineers, who reported that the time lost was negligible. The fact that the men know that every job is being watched, and the time taken thereon recorded, has had a marked effect, and cases have even occurred where men endeavoured to persuade the cost clerks to reduce the time taken on one job and charge it to the next, so that the engineer would not discover that they had taken an unduly long time over the first job.

Cases arise where the number of jobs upon which a workman is engaged in a day are too numerous to be separately recorded—*e.g.*, the man on the shearing and punching machine may be asked to shear or punch iron for fifty different jobs. Obviously, it would not pay to have the man change his tickets fifty times during the day. This difficulty is got over by having the man's time recorded against a "standing order," and the amount involved is regarded as an indirect expense, to be subsequently distributed with the other shop expenses. It has been made a fixed rule that no work of a construction or maintenance nature (except breakdown work occurring at night) may be started unless an order, signed by a duly authorized officer, shall first have been sent to the order clerk. Upon receipt of this order (called locally the "master card"), the order clerk issues copies of the order to the various foremen concerned. These orders constitute the foreman's authority for starting the work. The "master card" is filed in the workshops cost office on a large swing file, where it is readily accessible to any member of the cost office staff.

The names of the gangs to which copies of the order have been sent are recorded on the "master card," and upon completing their section of the work foremen are required to return their cards to the order office, where they are filed behind the "master card." Upon receipt of all foremen's cards the job is closed and its final cost reported.

Cases occur where, in order to complete his part of the work, a foreman may have to call in the assistance of another gang, the foreman of which has no order. In these cases foremen are permitted to issue sub-orders. These are in triplicate, one copy going to the cost office, one to the foreman who has to do the work, and the other being retained by the issuer.

Upon receipt of the job tickets from the men, the cost clerks proceed to post them under the appropriate job number to sheets, known locally as "Daily Labour Reports." Each gang is kept separate, and, to facilitate the conversion of hours into money at the end of the day, separate sheets are used for each rate. One group of cost clerks starts at 4 p.m. and works till midnight. They summarize the daily labour reports and post the totals to the "Job Cost Records," which show each day the cost of the work done by each gang, the total labour cost, the cost of the material used, and the total prime cost.

This particular section of the cost office work is kept closely up to date, and it is possible for the engineers each morning to ascertain the progressive cost to 5 p.m. of the previous evening of any maintenance or construction job which may be in progress. Details of all material used are also available if required.

This system allows a very close watch to be kept upon expenditure, and enables the engineers to check waste, both of labour and material. As each gang is kept separate, it is easy to fix the blame for inefficient work, and the slack foreman is readily detected.

Whilst, theoretically, all jobs should receive the same attention, it cannot be denied that the fullest benefit from the system is seen in jobs which are tendered for. Here the engineers have tied themselves to a definite figure, and are naturally anxious to keep within that limit. In these cases the figures are in daily demand, both for labour and material.

In addition to posting the figures to the "Job Cost Records," the cost clerks also prepare a summary of labour and material, classifying each under suitable headings, for use in connection with the costing of the various metallurgical operations.

Some time ago considerable confusion resulted owing to jobs being closed before all material used thereon had been debited up. This sometimes arose owing to invoices not being to hand. The store office is now required to send to the cost office daily a list of all requisitions which they are unable to price, and the job is not closed until all requisitions relating thereto have been dealt with.

OPERATING COSTS.

Departmental operating costs are compiled at four-weekly intervals for the following departments :—

Electric power.
Ore unloading and re-handling.
Dwight and Lloyd.
Dwight and Lloyd and H. and H. pots ("cross-over" process).
Smelting.
Refining.
Litharge making.
Skinner roasting.
Ropp roasting.

A feature of these cost sheets is the sectionizing of the department into operating units, the cost of operating each unit being separately shown. The refinery is a typical example, and is subdivided into—

Section A.—Copper dross furnaces.
,, B.—Antimony-softening furnaces.
,, C.—Desilverizing kettles.
,, D.—Refining furnaces.
,, Dx.—Crust enrichment.
,, E.—Market kettles and moulding pits.
,, F.—Gold alloy furnaces.
,, G.—Antimony dross furnaces.
,, H.—Retort furnaces.
,, I.—Cupellation furnaces.
,, J.—Remoulding, weighing, finishing, and granulating silver.
,, K.—Antimony blast furnaces.
,, L.—Dross re-handling and treatment.
,, M.—Weighing and disposal of finished products.
,, N.—General operating charges.

Each of these sections shows the cost of the following items :—

Foremen—operating.
Labour—operating.
Coal.
Supplies.
Power.

Maintenance.—Plant labour.
Shop labour.
Shop materials.
Direct stables service.

and wherever additional items are necessary, they are, of course, shown.

As indicated by section N above, there are certain operating charges which cannot be directly allocated to any of the preceding sections. These charges consist of :—

1. Salaries—Superintendent and assistants.
2. ,, Plant clerks.
3. Laboratory charge and sampling.
4. Light.
5. Water.
6. Locomotive service.
7. Fire service.
8. General labour and supplies.
9. Maintenance—Plant labour.
10. ,, Shop labour.
11. ,, Shop materials.
12. Proportion of indirect expenses from service departments.
13. Welfare expense.
14. Motor attendants and supplies.

As will be seen from a glance at the above list, many of the items are of altogether too general a nature to justify their allocation to any particular section or group of sections. For example, items 8, 9, 10, and 11 might apply to the repair of the refinery building, which covers all sections.

Item 12 may need some little explanation. The shop labour and material shown against sections A to M is the actual labour and material charged through the various sub-departments performing the service. Naturally, in each of these sub-departments there will be certain indirect expenses which are not directly applicable to individual jobs—*e.g.*, sweeping and cleaning shops, lighting of shops, power, time of foremen supervising a number of jobs, etc. For the purpose of the departmental costing it has not been thought worth while to allocate this class of indirect expense to the individual sections. It is considered that, from the point of view of an operating superintendent, a clearer view is obtained by showing the maintenance expense net, and including the whole of the indirect expense applicable thereto on a separate line.

A feature of the costing at these works is the distinction made between purely operating costs and the total costs applicable to a process. The former is held to be the only cost for which an operating superintendent can be held responsible. There are, however, certain general charges which are allocated between the various departments. These are shown on a separate sheet, and consist of such items as :—

1. Proportion of local indirect technical and office salaries.
2. Depreciation.
3. Insurance—fire.
4. ,, wages.
5. ,, consequential loss.
6. Experimental.
7. Rates, taxes, donations, legal, and all other administrative charges.
8. Attendance bonus.
9. Accident fund subsidy.
10. Foremen and bosses on sick or annual leave.

The unit used for costing in the various departments is as under : —

Electric power	Per kelvin.
Ore unloading and re-handling	Per ton.
Dwight and Lloyd ..	Per ton of material treated.
Cross-over	,, ,, ,, ,, ,,
Blast furnaces (smelting)	Per ton of charge.
Skinner furnace ..	Per ton of ore.
Ropp roasters	,, ,, ,, ,,
Lead refinery	Per ton of lead and antimonial metal produced.

and costs are recorded to one-tenth of a penny, except in the case of electric power, where the cost per kelvin is carried to three decimals of a penny.

In addition to the departmental costs, a final summary of all lead-smelting operations is compiled, giving the total cost at Port Pirie of producing one ton of lead.

To facilitate the collection of information, symbols are freely used to indicate departments and sections of departments. For instance,

A signifies Dwight and Lloyd department.
B ,, H. and H. or cross-over department.
C ,, ore and flux crushing department.
D ,, blast furnace department.
E ,, lead refinery department. etc.

and EA would refer to the copper dross section of the lead refinery department. These symbols are shown on all job orders. For example, the refinery superintendent might issue job order E/761, and would show that it was chargeable to EA5, which would mean that the work was in connection with the water service of the copper softeners. The cost of the job would be separately reported to him, but it would also be incorporated in the cost of that particular section of the refinery.

SERVICE DEPARTMENTS.

In a concern of any magnitude it will be found that, in addition to the main departments, there will exist a number of subsidiary departments, whose function it is to serve the main departments in various directions. These are known locally as "service departments," and they consist of—

Engineering workshops (mechanical and electrical).
Stables.
Store.
Wharf and shipping.
Sampling and assaying.
Locomotive traction.
Experiments and researches.
Accounting and general administrative.
Salvage and scrap.

Each of these will perform services for the main departments, and may also undertake work of a construction nature. Any one of them may also be called on to do some work for one of the other subsidiary departments. Work done by any of them for another department (whether operating, construction, or subsidiary) is charged direct as to prime cost—*i.e.*, labour and material. The allocation of the indirect expense from the service department to the job done by it is treated in the notes on overhead. The expense inseparable from the running of each service department is recorded in a separate ledger account, and is kept in such a way that the nature of each class of expense which goes to form the total can be clearly seen.

Overhead Expense.

This type of expense is also sometimes referred to as "Oncost," "Burden," "Indirect Charges," etc. Locally it is separated into the following groups :—

(*a*) Overhead directly applicable to operating departments—*e.g.*, salaries of the superintendents of departments and their assistants.

(*b*) Overhead directly applicable to the engineering department—*e.g.*, salary of chief engineer, draughtsmen, and draughting office expenses.

(*c*) Overhead directly applicable to service departments, as, for instance, the engineering workshops, which would cover the salary of the plant engineer and his assistant, foremen (when not on specific jobs), maintenance and repairs to machine tools, sweeping and cleaning shop, light, etc.

(*d*) Administrative expense, such as the salaries of the general superintendent, business manager, and accounting staff, wages of watchmen, chauffeurs, Red Cross attendants, sanitary men, rates, taxes, donations, legal, telegrams, postage, telephones, and all other expense of a general nature.

Class (*a*) is, of course, charged direct ; class (*b*) is charged out over the various sub-sections of the engineering department in proportion to the number of men engaged in each ; class (*c*) falls directly upon the particular subsidiary department concerned ; class (*d*), which, of course, represents the most important section, is distributed over the whole of the departments, both operating and service, in proportoin to the number of men engaged in each. It will thus be seen that, in the case of the engineering workshops, the overhead burden will consist of—

1. A proportion of the general administrative expense.

2. A proportion of the general engineering charges.
3. The whole of its own indirect expense, consisting of the salary of the supervising engineer, wages of foremen, maintenance of shop and tools, light, etc., and services performed for it by any of the other subsidiary departments.

The distribution of this indirect expense, of course, follows the direct labour.

STORES CHARGES.

Many of the costs of running the stores department cannot be charged to any specific article, as, for example, the salary of the chief storeman and the wages of the men handling stores, salaries of store clerks, freight on a number of small lines, etc. This class of expense is assembled in a ledger account entitled "Stores Charges," and is added as a percentage to the value of stores issued.

GENERAL.

Owing to the necessity of confining this paper within reasonable limits, it is obvious that many points of interest to accountants must be omitted, and no detailed mention can be made of such items as the tabular cash book in use, introduced with the idea of condensing information for transmission to head office; inter-departmental debit notes, for ensuring that none of the numerous charges against outsiders for services rendered shall be overlooked; mechanical means of recording truck weights; records connected with the sick and accident fund and the co-operative store; condensed metallurgical records, special form of tabular general ledger, method of reconciling costs against ledger accounts, etc.

DETERMINATION OF FINENESS OF SILVER BY VOLHARD'S METHOD, AS PRACTISED AT THE PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

By T. A. Keats.

This excellent volumetric method commends itself as a means of estimating the fineness of silver—firstly, owing to the large number of possible impurities which do not affect its accuracy (these impurities include Cu—up to 70 %—Au, Pb, Zn, Bi, Cd, Fe, Mn, Sb, and As); secondly, the method combines accuracy with speed.

The above impurities are the only ones likely to be present in the B. H. A. S. Pty. Ltd. refined silver, and, with the exception of Cu and Au, the others, if present at all, would be well under 0.1 parts per 1000.

Briefly stated, the method consists in the solution of the refined silver in HNO_3, addition of ferric alum (indicator), and titration with an alkaline (either ammonium or potassium) thiocyanate solution until the brown ferric thiocyanate is formed, and gives a definite tint to the solution. The method is used at the Port Pirie smelters, and the outstanding feature in the method as there applied is the type of burette employed. This burette enables the operator to read 0.01 of a cc. with ease, whereas, with the ordinary burette fitted with a float, 0.05 of a cc., when a number of readings have to be taken, becomes very straining to the eyes. Moreover, waiting for the float to rise, and dislodging air bubbles from it, hinders rapid working. A description of the burette and fittings is appended with fig. 2.

Details of the Method.

Solutions required are :—

(*a*) *Stock Solutions.*—

(1) Ammonium thiocyanate, 40 grm. per litre of water; 30 litres are prepared and placed in a carboy with glass syphon attached.

(2) Ferric alum, 250 grm. per litre; two litres are prepared.

(*b*) *Working Solutions.*—

(1) Ammonium thiocyanate.—930 cc. of stock solution are syphoned off and made up to 5 litres with water. Thirty litres are prepared and placed in a carboy fitted with a glass syphon. The strength of this solution is tested immediately. One grm. of standard silver is titrated, and should require 99.75 cc. or thereabouts. This ensures the refined silver titration being completed at about the middle of the capillary tube on the burette. Some slight adjustment of the solution may be required, but a little practice enables one to do this readily.

(2) Ferric alum (indicator).—One litre of stock solution added to 4 litres of water.

(3) Nitric acid.—Two litres HNO_3 (1.4 sp. gr.) to 1.5 litres of water.

The working solution of ammonium thiocyanate is stirred well with a rod every day before using.

Standard Silver.—This has a fineness of 999.4, and is granulated.

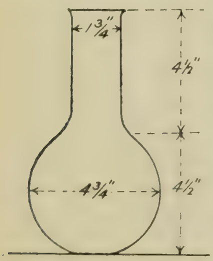


Fig. 1.

Standards.—Weigh out three lots of 1 grm. of standard silver; place in a 700-cc. flask (see Fig. 1). Add 5 cc. HNO_3 solution, place the flask on a hot plate, and heat until all brown fumes are driven off and silver dissolved. Remove and allow to cool for at least 15 minutes, add 100 cc. of cold water and 5 cc. ferric alum indicator. The standards are now ready for titration.

Titration.—Fill the burette with the working solution of ammonium thiocyanate and run out; repeat this four or five times to ensure that the burette is clean. Fill the burette and remove all air bubbles by raising the jet of the valve and allowing a little of the solution to run out. Finally adjust the solution to the mark on the top capillary tube. Run the solution into the flask (keeping the flask agitated all the time) to the 99.75 mark. Remove all solution from the end of the burette by tapping gently three or four times on the neck of the flask, and rinse the neck of the flask with cold water from a wash bottle. Now shake the flask vigorously 200 times, again rinse the neck of the flask, and read the burette. The time taken for shaking (about 2 minutes) is sufficient to enable the solution in the burette to drain thoroughly.

A bracket at a convenient height, and long enough to take three flasks abreast, serves to hold the flasks when titration is completed. A piece of white paper is placed as background, and enables one to distinguish the brown tint readily. Three standards are done, and their colours compared. Two of these must agree before accepting one as the standard.

Precautions.—

(1) The wash water should be about the same amount for each assay, and as little as possible.

(2) The time allowed for draining of the solution in the burette should be sufficient.

Calculation.—

$$\begin{aligned} \text{Standard silver used} &= 99.94\ \% \text{ or } 999.4 \text{ fine} \\ \text{Assume reading of burette} &= 99.70 \text{ cc.} \\ \hline \text{Difference} &= 0.24 \end{aligned}$$

This shows the solution to be slightly stronger than 1 cc. = 0.01 grm. of silver, and that it is necessary to add 0.24 to the readings of the burette to obtain % of silver. Moving the decimal point one place to the right gives fineness of silver or parts per 1000.

Refined Silver.—Take 1 grm. of sample (granulated), place in a 700-cc. flask, and proceed in the same manner as for standard. Carefully add the last few tenths, so that the final brown tint obtained agrees with the tint produced in the standard. To the burette reading add 0.24 and move the decimal point one place to the right, which gives parts per 1000 or fineness of the sample.

Burette and Fittings.—That portion of Fig. 2 above A is of special construction, and might be termed the burette proper. The length of this portion is 1 ft. 2 in., and consists of the two capillary tubes AB and CD and the sphere E. The capacity of that portion of the burette B to C is exactly 99 cc. B to A holds the remaining cubic centimeter, and is the only part which is graduated. It is divided into tenths, and these again into tenths, thus marking one-hundredth of a cc. One-tenth of a cc. in this capillary tube lowers the column 0.7 of an inch, so that the one-hundredth divisions are about 0.07 in. apart, and are thus easily read off.

Below A are the fittings, which consist of rubber tubing F, T-piece K, taps L and M, glass valve, and jet O. One end of the T-piece communicates to the working solution of ammonium thiocyanate, the supply of which is controlled by taps L and M. To the other end of the T-piece is fixed the glass valve, which consists of a piece of good rubber tubing into which is fitted tightly a piece of glass rod about ½-in. long. The ends of the glass rod are tapered slightly and rounded off. To the other end of this rubber tubing is attached the jet, which is similar to the ordinary wash-bottle jet. A gentle pressure on the glass rod which is in the rubber tubing allows the solution to run out.

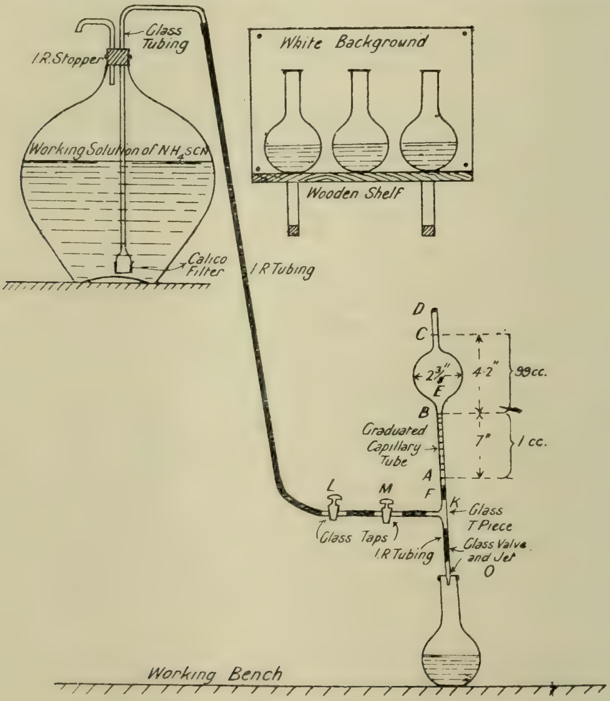


Fig. 2.

Burette and Fittings.

MECHANICAL DEVICES ORIGINATED AND USED AT THE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD., PORT PIRIE, S.A.

BY. J. AHERN.

THE writer of this paper well realizes that the majority of engineering and metallurgical works have from time to time evolved mechanical devices which are of great value as regards application in an operating plant.

Many of these devices would not be apparent to an individual making a brief inspection of works such as these, and it is for this reason that the writer proposes to outline a few of the devices which have been evolved by the engineering department of the Broken Hill Associated Smelters Pty. Ltd.

NOVEL METHOD OF MACHINING A SPHERICAL JOINT.

Fig. 1 represents one half of a casing for a ball-joint coupling. In the ordinary methods of boring an external spherical casing, that is with a radius bar attached to the tool rest of a lathe, there is always an error in the bore. It has two flat sides, and this error is inherent in the ordinary orthodox method and cannot be corrected.

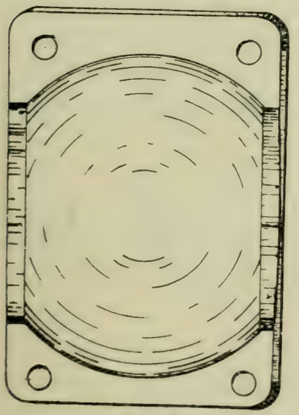


FIG. 1.
HALF OF SPHERICAL CASING.

The engineers of the Broken Hill Associated Smelters Pty. Ltd. undertook a job that had to be extremely accurate, and the tool shown in Fig. 2 was evolved. The tool is not exactly fool proof, but in the hands of an average mechanic an absolutely perfect sphere can be bored. The casing has to be held in the lathe in such a manner that one half can be removed without affecting the setting of the work. The business end of the tool is then placed inside the remaining half case and the shank is gripped in the slide rest of the lathe. The tool point must be set exactly on the vertical centre-line of the lathe and the revolving disc set central. The half casing that has been removed can now be replaced on the job. By rotating the handle the disc is revolved on the central pin and the inserted tool will cut an exact sphere, the diameter of which can be varied by moving the cutting point outward until the correct size is obtained.

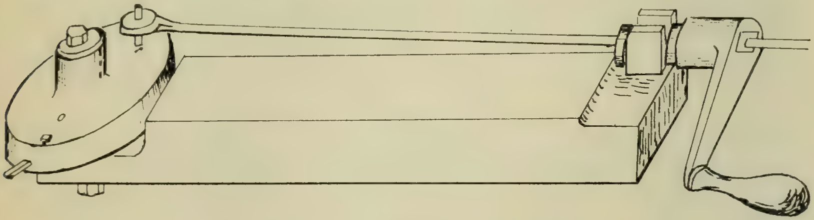


Fig. 2.
Device for Boring Spherical Casing.

Reinforced Castings.

A deal of trouble was once experienced with the cast-iron pallets on the Dwight & Lloyd machines. Owing to the hard knocks they receive when they go over the tip end of the machine they frequently fracture and parts of the pallets find their way into the sinter crushers and cause a lot of damage. This trouble has been almost entirely eliminated by placing a mild-steel bar through the mould in which the pallet is cast and running the metal around it. This method of reinforcing castings is shown in Fig. 3.

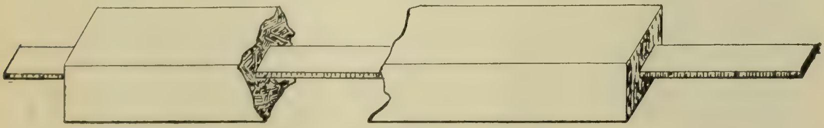


Fig. 3.
Method of Reinforcing Cast Iron with Wrought Iron Bar.

It is not claimed that any additional strength is given by the use of the reinforcing bar, but it is claimed that the trouble was cured. The pallets can crack now in three or four places but they are held together like a lot of beads are held on a string, and go on with their work as well as ever.

There are a few points in connection with this that are worth knowing. It is essential that all moulds are thoroughly dry before being poured, because if damp sand is used, drops of moisture form on the reinforcing bars and blow-holes are the result. The most successful results seem to be obtained when the relative proportions of the cast-iron and the reinforcing bar are about 5 to 1. Clean bar stock must be used for reinforcing. If rusty, it must be put through a rumbler.

ELEVATOR BUCKET.

Fig. 4 represents an improved type of elevator bucket. It was found that the ordinary buckets elevating solid material, viz., granulated slag, from under water, were simply draining from one bucket to another. One of the engineers, Mr. Tonkin, made and installed buckets shown in Fig. 4. The projecting lip at each end should be noted. This idea effectually cured the whole bother and incidentally reduced the load on the motor.

CAST-IRON WATER JACKETS.

Fig 5 represents one of the water jackets. These jackets are standardized. They have 5 studs and 2 water-pipe connections in each.

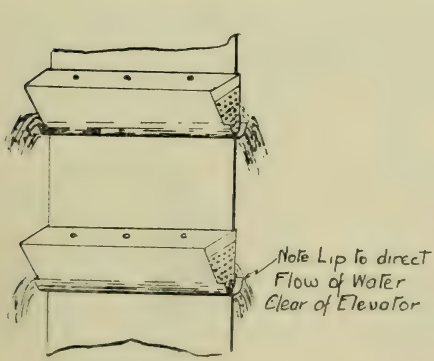


FIG. 4.
ELEVATOR BUCKET.

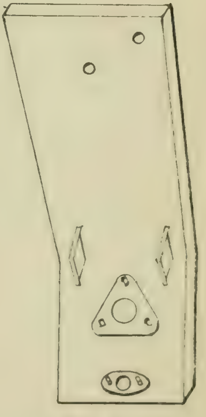


FIG. 5.
STUDS AND SOCKETS CAST IN WATER JACKETS.

Two of the studs are for the cleaning door, the other three are to hold the tuyere casting. The water connections are for 1½-in. pipe and are for inlet and outlet of the cooling water. The jackets weigh about

7 cwt. each, and at one time lumbered up the workshop very considerably, as it was the practice for a long time to drill and tap the stud and pipe holes. For some time past these stud and pipe connections have been cast in the jacket and have saved endless trouble. The jackets are never inside of the workshops now.

At first a little bother was experienced with the stud and pipe sockets. If they were twisted too hard they would loosen. A recess is now cut in the part that is cast in the jacket, and this effectually locks them. They can neither back in or out of the casting. These recesses are shown in Figs. 6 and 7.

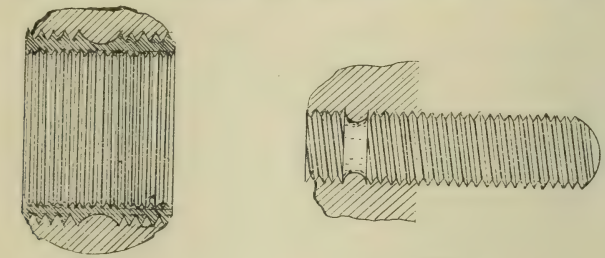


Fig. 6. Fig. 7.

Stud and Socket showing Lock Recess.

Tuyere Casting.

Fig. 8 shows the poke-hole end of a tuyere, with cap removed. The cap is run on a fast thread $\frac{3}{4}$ in. pitch, and this thread, and also the female thread in the cap, are very successfully cut by the moulder in the sand.

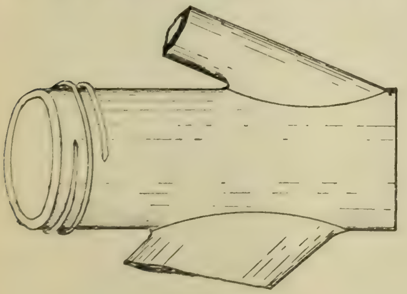


Fig. 8.

Thread Cast Solid on Tuyere to Obviate Screw Cutting.

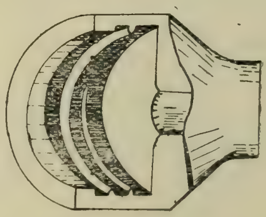


Fig. 9.

Half of Core Box for Casting Threads in Caps.

Fig. 9 is a view of one half of the core box used.

MANDREL FOR USE IN RUNNING BEARINGS.

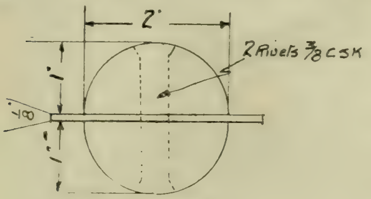


FIG. 10.
END VIEW OF MANDRIL.

Fig 10 represents a mandrel used for white-metalling bearings. The fin in the centre is made to fit between the cheeks of the bearing and the cap is then bolted down in place. Both top and bottom halves are then run in one pour. The mandrel is then removed and no further work, except cutting oil races, should have to be done on the bearing. This type of mandrel in all sizes of bearings from $1\frac{1}{2}$ in. to 4 in. is used, and no trouble is experienced with it.

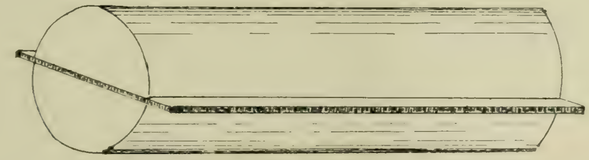


FIG. 11.
ELLIPTICAL MANDRIL FOR RUNNING UP WHITE METAL BEARINGS.

USEFUL MODIFICATION OF EMPTYING COCK ON OIL DRUMS.

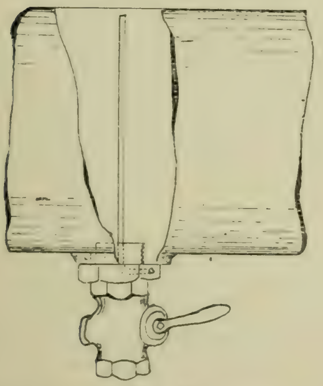


FIG. 12.
TAP WITH AIR VENT TO ELIMINATE PUNCTURING DRUMS.

Fig. 12 shows how a troublesome job was saved by one of the engineers, Mr. Tonkin. The works used hundreds of drums of crude oil. These drums had one screwed connection on them for filling and emptying, A vent hole had to be punched in the drums and subsequently soldered up. Fig. 12 illustrates the method of admitting air without a vent hole. A small tube rivetted in the emptying cock and projecting up to the top of the drum effectually solved the problem.

OXY-ACETYLENE WELDING AND CUTTING OF ALL DESCRIPTIONS OF METALS—PRACTICE AT PORT PIRIE WORKS OF THE BROKEN HILL ASSOCIATED SMELTERS PTY. LTD.

By W. TONKIN.

OXY-ACETYLENE system of welding and cutting is at present practised in two forms, which may be described as the high and low-pressure systems.

In the high-pressure system the two gases, oxygen and acetylene, are delivered to the blow-pipe under pressure from properly constructed trade cylinders. Oxygen cylinders are in two sizes, each containing 100 and 200 cub. ft. of gas respectively. These cylinders are charged to a pressure of 120 atmospheres, or 1800 lb. per sq. in. approximately. The acetylene cylinders are made in one size as far as the writer knows, and contain 200 cub. ft. of acetylene gas at a pressure of 10 atmospheres.

Each cylinder is provided with a valve and connection for attaching a regulator to "break down" the high pressure in the cylinders to the necessary low working pressure of the blow-pipes, and to deliver the two gases at a constant pressure irrespective of the variable pressures in the cylinders. These regulators can be adjusted down to as low as ½ lb. and are provided with two pressure gauges—one reading the contents of the cylinder in atmospheres, and the other the working pressure in lb. per sq. in.

The high-pressure, or "two cylinder" system, is compact and can readily be transported from place to place in the works. For this purpose the two cylinders are mounted on an easy-running 4-wheeled truck, which the operator and assistant find no difficulty in manipulating.

In the low-pressure system the two gases are delivered to the blow-pipe, one from the oxygen cylinder and the other from the acetylene gas generating plant, which consists of the following:—

Generating cylinder, half filled wtth water, into which the carbide is plunged. The generated gas is passed through a filter to the gasometer, then through a water washer on to the hydraulic seal which acts, in this case, as a safety valve to prevent any back-firing from the jet getting into the gasometer. The pressure-gauge for the oxygen is operated in the same manner as in the high-pressure system. The acetylene pressure is obtained by weighting down the gasometer "bell" to the required pressure. This is about 8 in. of water. This system is well suited to shop work where the apparatus can be permanently set up in the one position.

The temperature generated in the blow-pipe flame is approximately 6000° F.

Autogenous welds can be effected by the oxy-acetylene blow-pipe without any injurious effect upon the metal. The flame is short and steady, and under good control. Constant practice is required to make workmen proficient in the manipulation of the blow-pipe, and the tradesman is trained on the simpler jobs, graduating as his experience grows.

The strength of the weld produced by this system is slightly less than the strength of the original material. This may be due to the use of welding strips of inferior tensile strength, or to other internal strains which are set up by the expansion and contraction of the work whilst being heated and cooled off.

Much depends on the care and intelligence exercised by the workman in properly preparing, cleaning, and fitting his work, preheating where necessary, and making due allowances for expansion and contraction, which is a very important factor.

Cutting iron and steel is very simple, and can be mastered by the workman in a very short time. The edge or surface of the plate at the point to be cut is first heated by the mixed jet of oxygen and acetylene. When this spot has been brought to a state of incandescence, a fine cutting jet of oxygen is discharged upon it. This immediately produces combustion of the metal with the resulting formation of iron oxide. The jet of oxygen is made sufficiently strong to blow away this iron oxide in front of it, with the result that a clean narrow cut is effected through the metal at a speed of travel which varies according to the thickness of the metal to be cut. The Broken Hill Associated Smelters Pty. Ltd. use this tool very extensively in cutting girders, steel plates and all other heavy work that could not be taken to the saw, punching, or shearing machinery. In the works it is absolutely indispensible.

Cast-Iron Cutting by Oxy-Acetylene—The writer believes that prior to the Broken Hill Associated Smelters Pty. Ltd. doing it in September 1919, cast-iron had not been successfully cut by the oxy flame. Numerous enquiries throughout the Commonwealth and perusal of all the available literature on the subject, indicated that it was not considered practicable. In fact one book said it was a waste of time to try.

The writer decided to try out experiments on his own account, and in a very short time discovered a flux which enabled him to cut cast-iron without any difficulty, and he feels sure that this is the one and only method of doing it. This process has been in constant operation from the above-named date and has given excellent results. The writer claims to be able to cut cast-iron with as much ease as steel-plate, but the cost is about 2½ times that of steel-plate cutting.

There are no sparks upward whatever, and the ordinary goggles give all needed protection. It is a clean and continuous blow through the metal.

The process is very simple and is within the power of any skilled operator. It is essential to get the plate, or article to be cut, heated to a state of incandescence with the oxy-acetylene jet, the same as is done when cutting steel. When this stage is reached, release the central oxygen jet and then start puddling and adding pure iron about ⅛ in. diam. into the cut. This acts as a flux, and being mixed with the cast-iron allows it to be freely blown through with the jet. It also acts as a decarboniser and overcomes the spots of graphite that appear so frequently.

The following list gives some indication of the extent to which the oxy flame is used here :—

(1) Brazing of all descriptions of copper and brass pipes which is, in all respects, superior to the open-fire brazing.

(2) Welding or building up worn parts in steel, cast-iron, copper, and brass. This is very extensively carried out and saves a lot of money—obviating the purchase of new parts.

(3) Welding of sheet-steel tubing in preference to rivetting in certain cases.

(4) Welding broken cast-iron grate-bars from the Dwight & Lloyd plant, which has been very successfully done.

(5) Welding new ends on B & W boiler tubes. Some time ago a special test was made of a boiler tube about 18 ft. long and 4 in. in diam. The welded joint was about 3 ft. from one end. To test this hydraulically both ends were welded up, a $\frac{3}{4}$-in. pipe tapped in one end for connection to the press, and the tube tested up to a pressure of 1 ton per sq. in., without indicating any weakness or flaw in the workmanship.

The above are but a few of the purposes for which the oxy flame is used, but to put it briefly, the writer can confidently say that there is hardly a job which comes into the shop where they cannot, with advantage, put the oxy torch into operation.

PROCEEDINGS AUS. I.M.M. (INC.)] [N. S., No. 47, 1922.

CONSTRUCTION COSTS.

BY G. McI. HUNTER AND L. CERUTTY.

To those who have been engaged in the construction of mining and metallurgical plants during the war and post-war periods, it is quite unnecessary to point out the large increases and violent fluctuations in prices of basic commodities and the steady and persistent rise in labour costs. It may, however, be of interest to present these facts in graphic form, and survey the relative heights of the various maxima and the lead and lag of individual waves.

Since Descartes demonstrated for us moderns the analogy between geometric form and algebraic statements, the two-dimensional graph has found widespread application. Mathematicians have developed for our information, if not for our confusion, the triangular diagram and the solid three-dimensional figures, which have proved so useful, at least in the hands of the metallurgists at Port Pirie.

The model (Plate I.) which is exhibited here does not pretend, however, to indicate the connection between any three inter-related variables, and it is not proposed to burden members with any abstruse mathematical expression for the irregular surface of the blocks. While time is an ordinate common to all sections of the model, each illustrates separately the variation in cost of some particular quantity between the years 1914 and 1922. It is an easy matter to record such individual variations, but it would be an impossible task to state any equation describing accurately the dependence of the prices, say, of hardwood or of galvanized iron upon that of labour at any particular point of time. The cost of living curve, which has attained such prominence, if not notoriety, in the last few years, is the result of an attempt to show the mass-effect of the components. It is the difficulty found in assigning the correct weight to the separate items which makes the aggregate curve somewhat misleading at times.

Turning to the model, it will be noticed that each section consists of half a dozen pegs. The first in each row, that coloured red, represents the cost of some particular quantity, such as bricks, timber or coal, at Broken Hill in 1914. The next peg, that coloured blue, represents proportionately prices for that quantity in Hobart during the same year. The next four pegs in order show the variation in prices for the ensuing years 1916, 1918, 1920 and 1922. Broken Hill data were taken as basic figures since definite information for that mining field was available. In some cases it will be noted, the authors have been obliged to commence with a Tasmanian figure as a base.

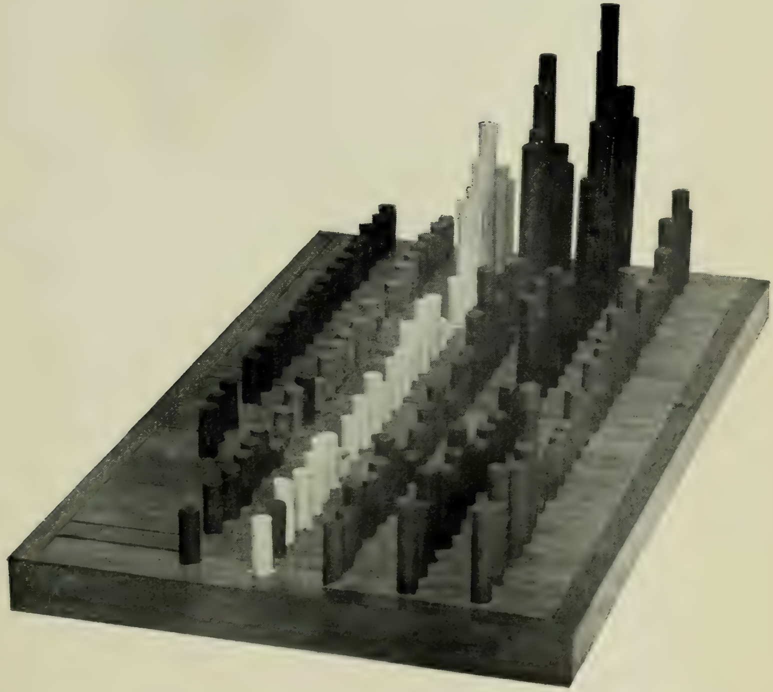
It was thought desirable, in the interests of clearness, to express the variation in prices as percentages of some standard, rather than to show actual figures, and to make the base pegs uniform in height. This arrangement makes only one vertical scale necessary, that of percentages. For the rest, the model explains itself and indicates clearly how costs were inflated, particularly after the cessation of hostilities. Even to-day it will be noted that the prices, as indicated by the last row of pegs, are still considerably above those of 1914.

To further illustrate the rocket-like elevation of building costs, the data on which the model is based have been used to estimate the cost of a hypothetical plant at various periods. The building taken as a type is considered to have a wooden frame covered with galvanized iron, concrete floors, a considerable amount of brickwork, steel work and extra items such as electric motors, fans, water and compressed-air service. Prices of labour and material are taken as those in force in the three years mentioned. For the same plant the costs become :—

1914, £12,377 ; 1920, £25,185 ; 1922, £22,710.

To quote several other detailed examples, the following tabulation indicates the variations in the prices of certain specific items —

	Broken Hill 1914.	Hobart.				
		1914.	1916.	1918.	1920.	1922.
	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.	£ s. d.
Galz.-iron sheet per ton	22 0 0	20 0 0	37 0 0	45 0 0	40 0 0	22 0 0
Steel plates per ton	12 0 0	11 5 0	36 0 0	50 0 0	62 0 0	36 0 0
Cement per ton ..	6 5 0	4 5 0	4 5 0	5 5 0	10 5 0	8 5 0
Red bricks per 1000	3 17 0	2 15 0	3 0 0	3 0 0	3 10 0	3 18 0
Average tradesman rate per day ..		11 0	12 0	13 4	16 5	1 0 3



Reading from left to right the rows are coloured—Red, Blue, White, Brown, Black, Green.

Quite recently an old estimate prepared in England in the year 1872 came under notice. Cast-iron at £13 10s. per ton, iron forgings at £32, brass fittings for a fan at 1s. 3d. per lb. formed an interesting comparison between prices obtaining in the post-war period of the early seventies and those of to-day.

Several points of interest appear when the result of several tables are thus brought together in a model in visible form. The lag of labour costs behind that of material is noteworthy. With most supplies the maximum is reached at the fifth peg; labour cost more in 1922 than in 1920. It is this lag that is apt to be a cause of unrest to-day.

As has been noted, the pegs illustrating conditions in 1922 are still a good deal higher than the original 1914 standards. We certainly are a considerable distance away from President Harding's "normalcy," and he would be a bold prophet who would venture to-day to whittle out suitable pegs for 1924.

A MECHANICAL DISINTEGRATOR FOR SLIME SAMPLES.

By T. A. Read.

(*Read before the Broken Hill Branch.*)

On milling plants, where a large number of slime samples are collected each day, considerable work is involved preparing them for the assayer.

The usual practice (as the material usually passes a 150-mesh screen) is to place the sample on a 30-mesh screen and brush it through on to a cloth below. The sample on the cloth is mixed and a portion taken out for the assayer.

A simple apparatus that would rapidly disintegrate and prepare these slime samples for the assayer was needed. The manufacturers of assayers' supplies appear to have overlooked apparatus for this class of work.

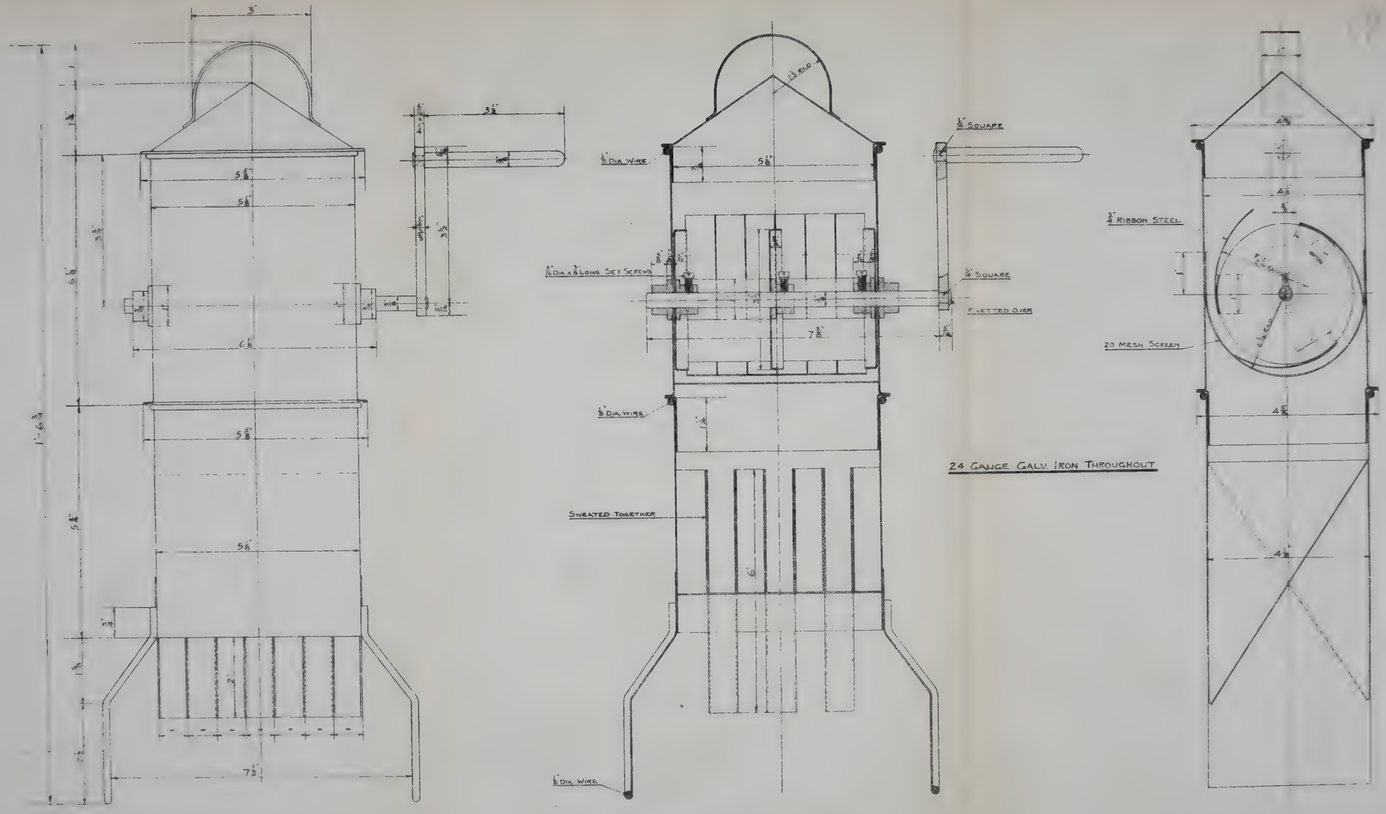
To overcome the tedious preparation of slime samples by the above method, the writer devised the apparatus shown in the accompanying drawing (Fig. 1). It will be seen that the apparatus is a combination of a modified household flour sifter and a riffle sampler.

The apparatus is made up in two parts, the top half consisting of a screen box carrying the disintegrator, and the bottom half a riffle divider, the joints between the two being a slip-in fit. A lid is provided for the screen box.

The disintegrator consists of an open drum, to which are soldered three rows of spring steel fingers. Each row has a number of steel fingers which, on rotation, press against the semi-circular screen in the screen box, forcing the slime particles through.

To operate the apparatus an 8-in. sample tin is placed under each spout of the divider. The sample of slime, which is about 2 lb. in weight and usually in lumps, is placed in the screen box, the cover replaced, and the drum rotated about half-a-dozen times by means of the handle.

The disintegrated sample is divided by the rifle divider into two portions, which are found in the tins under the spouts of the apparatus. One of these samples is taken, mixed on a sheet, and sent to the assayer.



ARRANGEMENT OF DISINTEGRATING SAMPLER.
SCALE—⅓ FULL SIZE.

NOTES ON THE ESTIMATION OF TUNGSTEN IN ORES.

BY J. R. POUND, M.Sc.

THE following notes simply sum up the writer's experience, which extended over a very busy period of some three years, dealing with wolfram ores and concentrates. The three methods for the estimation of tungsten that are discussed below are well known, but these results of their trials and comparison under technical conditions may be interesting.

Method 1.—The routine procedure for the assay of rich wolfram ores or concentrates was as follows :—Take 1 gram of finely-ground sample (1) ; digest with HCl and HNO_3 for about 2 hours, evaporating down to *near* dryness (2). To the remaining 5 cc., say, of liquid, add water, warm or boil for ¼ to ½ hour, filter through a small filter paper, and wash the beaker and the insoluble matter with hot dilute HCl. It is best to keep the bulk of the insoluble matter (WO_3, SiO_2, etc.) in the beaker at this stage. Then place a weighed porcelain dish under the filter funnel and wash the beaker and the filter paper with successive small quantities of strong NH_4OH, finally bringing all the insoluble matter on to the filter paper, and collecting the ammoniacal filtrates in the porcelain dish (3). Then place the filter paper with the insoluble matter in a porcelain crucible, dry and burn off the paper, cool, and brush the contents of the crucible on to an agate mortar, grind it to flour, and brush this back to the original beaker without loss. In the absence of draughts this can be done safely. Re-treat this residue with acid and ammonia as above, running the second ammoniacal filtrate into the previous porcelain dish, which in the meantime has been heating on a water-bath (4). Finally evaporate to dryness on the water-bath the combined ammoniacal filtrate, and then heat gradually to a red heat over a Bunsen burner, cool, and weigh. Then the contents of the porcelain dish give the WO_3 (tungstic acid) directly (5).

The final filter paper and insoluble matter may be ignited in a porcelain crucible, thus giving the "insoluble matter." If the ore is only wolfram with insoluble matter, which is often the case, then $(\%\ WO_3 \times \frac{4}{3} + \%\ \text{insol.})$ should reach 100 %, and this is sometimes a useful check. Pure wolfram (FeMn) WO_4, contains about 76 % WO_3. Some authorities quote a maximum value of 76.6 % WO_3, but the Tasmanian wolfram best known to the writer contained at the most 75.8 % WO_3.

Notes on Method 1.—(1) The attack of HCl or *aqua regia* on wolfram produces the insoluble WO_3, and thus the action is slow, and the last portion of the wolfram is liable to be protected from decomposition. Fine grinding is essential, therefore, and especially for material containing over 38 per cent. of WO_3—*i.e.*, over 50 per cent. of wolfram. The actual assay sample of such material must be ground to flour in an agate mortar. This is easier said than done. Also, during the acid treatment the sample is liable to "crust up" and thus to protect the lower layers from the action; this crust should be broken up several times during the action. When products containing less than 50 % wolfram are ground, and the wolfram is the softest constituent, then a ground product is easily obtained in which all the wolfram is fine enough to be completely and readily decomposed by acids; and if the other constituents of the ore are insoluble in the acids, their presence keeps the wolfram particles separated, and considerably helps the complete decomposition of the wolfram, and in this case the second acid and ammonia treatment is unnecessary Such common constituents of wolfram ores are silica, silicates, and cassiterite.

(2) The writer could see little if any advantage accruing from prolonged action of HCl alone; therefore, after treatment with HCl for about $\frac{1}{4}$ hour the HNO_3 was added and the solution slowly evaporated down. The usual amounts of acids were about 40 cc. HCl and 5 cc. HNO_3, and the time of treatment would vary from 1 to 3 hours. It is doubtful if prolonged action of acid is worth the time —*i.e.*, probably the bulk of the wolfram is decomposed within an hour. The second acid treatment, necessitated by the imperfect grinding or the incomplete decomposition in the first case, is completed in a short time—say, $\frac{1}{2}$ hour. Evaporation of the acid right to dryness is liable to render the WO_3 difficultly soluble in ammonia.

(3) It is necessary to wash the WO_3 on the filter paper with strong ammonia, otherwise colloidal silica will run through and ultimately be weighed as WO_3. It is not necessary to add ammonium chloride to the ammonia wash liquor, and the presence of excessive ammonium chloride with the ammonium tungstate in the porcelain dish causes the contents to flake and jump off the dish during the ignition, thus producing mechanical loss of WO_3. The ammonia washing of the WO_3 sometimes becomes slow; possibly the silica (or silicic acid) clogs the pores of the filter paper. It is thus best to bring the bulk of the insoluble matter on to the filter paper only with the last ammonia washes. The ammonia filtrate should fit comfortably in an 80-cc. porcelain dish.

(4) The use of a water bath is recommended, as the ammoniacal liquor then evaporates to dryness quietly and safely, and needs no watching.

(5) In this method there is a tendency for colloidal silica to run through with the ammonia liquor, but this may be kept at a minimum by the conditions described under note (3). If the ammoniacal filtrates are collected in a platinum dish the SiO_2 may be removed by the usual H_2SO_4 and HF treatment, and this treatment might be necessary for low-grade material containing easily decomposable silicates; it was not adopted in the work covered by these notes.

The filtrate after the first acid treatment can be tested, after dilution, with H_2S, and thus CuS, Bi_2S_3, etc., can be precipitated and qualitatively and quantitatively estimated, and afterwards, of course, other constituents of the ore—*e.g.*, the calcium from scheelite, $CaWO_4$—can be detected and estimated in the usual way.

The final insoluble matter may be inspected for silica or rock minerals (white), titaniferous iron ore (black), and especially for cassiterite (after the acid treatment, brownish). Such an inspection, after a little experience, will enable the tin content of the wolfram concentrates to be estimated to the nearest per cent., especially when the amount is small—say less than 5 % Sn. In any case the tin content of the sample may be obtained with accuracy by Pearce's method applied to this insoluble matter—*i.e.*, by fusing it with Na_2O_2, dissolving the melt in HCl, reducing the strongly-acid hot solution with nickel, cooling in CO_2, and diluting and titrating the $SnCl_2$ solution with an iodine solution which has been standardized similarly against pure metallic tin. The writer found this to be the most satisfactory method of estimating the tin in all cassiterite-wolfram concentrates.

When dealing with or purchasing various and unfamiliar wolfram concentrates, the above acid method thus enables the other constituents to be detected and estimated, if necessary, with the minimum of trouble, and this is an important feature. Nevertheless, as will be seen later, with concentrates rich in wolfram care is necessary to avoid low results for the tungstic acid content, and in general the higher of two duplicates will give the more correct per cent. WO_3. The common tendency is for some of the wolfram to remain unattacked or for some of the WO_3 to remain occluded as ammonium tungstate in the filter paper. In the latter case the yellow WO_3 will be seen in the final ignited insoluble matter. By this method duplicate results should agree to within 1 % of the wolfram content of the sample.

Method 2.—A modification of the above method, which obviates the last-mentioned source of error, is to grind *perfectly* at the start so as to ensure complete decomposition of the ore in one acid treatment, and, after washing the WO_3 free from acid, to place it, filter paper and all, in AmOH solution in a graduated flask or beaker (the original beaker may serve), and after its dissolution as ammonium tungstate to make up to the definite volume, and then to take of the clear supernatant liquid a definite aliquot part, which, on evaporation to dryness and ignition, gives the WO_3, and hence the % WO_3 of the sample is calculated. This method does not necessarily demand complete disintegration of the ore in one acid treatment, but such would be advisable. Excepting that the remaining insoluble matter is not directly collected, this modified acid method is perhaps better than the previous one—*i.e.*, it is inclined to give higher results for the % WO_3.

Method 3.—The writer also used the well-known mercurous nitrate method of estimating tungsten, and by this means correct results may be obtained. This method was carried out as follows:—Take ½ gram of wolfram concentrates and fuse with NaOH in a nickel dish for 15 minutes; then dissolve the melt in hot water, a little Na_2O_2 being added, and then filter hot from the insoluble Fe and Mn oxides, unattacked SnO_2, etc., and wash well with hot water. Then slightly acidify the filtrate with HNO_3, heat till the CO_2 escapes and the separated silicic acid (and stannic acid) is coagulated, and then make alkaline with ammonia, heat and filter, and wash. Again slightly acidify the filtrate with HNO_3, and precipitate the tungsten as mercurous tungstate, Hg_2WO_4, a pale yellow coagulable precipitate, by the addition of $Hg_2(NO_3)_2$ solution. This last step demands a nearly neutral solution, but as the $Hg_2(NO_3)_2$ solution can only be made with excess of HNO_3, the assay solution becomes more acid during the precipitation; small excess of $Hg_2(NO_3)_2$ is added and then dilute AmOH drop by drop till the liquor is neutral or slightly acid, when all the tungsten is precipitated as Hg_2WO_4, along with more or less black Hg and mercury amino-compounds. As Hg_2WO_4 is soluble (with decomposition) in excess of AmOH or of HNO_3, the conditions of precipitation must be carefully observed, otherwise it is possible to leave tungsten in the solution. The tungsten bearing precipitate is well washed with hot water containing a little $Hg_2(NO_3)_2$, the precipitate is dried, placed in a porcelain crucible, and ignited, and the resulting WO_3 is weighed.

This is probably, when combined with suitable modifications (*vide* note "5" above), the most accurate method of separation of the tungsten from all other constituents of most wolfram ores, e.g.—SiO_2, SnO_2, Nb, and Ta; but it is not suitable for scheelite or for ores containing Ca and similar metals, appreciable metallic sulphides, especially arsenical pyrites, or phosphates, in which cases the acid method is preferable. In the $Hg_2(NO_3)_2$ method the "fiddling about" necessary to ensure the complete precipitation of the wolfram takes time, and also the other constituents of the original ore are not separated in a form in which they can be readily detected or estimated, and finally this method is rather expensive. These facts, and the almost general adoption of the acid method by others (buyers or sellers), led to the restricted use of the $Hg_2(NO_3)_2$ method.

The following table will indicate some results on wolfram ores or concentrates obtained by means of these three methods. The figures quoted are in all cases the means of duplicates. The results by methods (1) and (3) were obtained by the writer on the same bulk assay sample; the results by method (2) were obtained by an independent assayer on his own sample, and thus between (1) or (3) and (2) there is a possible divergence due to the two samplings, but as these were carried out on approximately 5-ton parcels of the fine material (minus 6 mesh), shipped in bags of 1 cwt. capacity, the possible errors of sampling should be very small.

Percentage weight of tungstic acid (WO_3) in wolfram ores as determined by—

—	(1.) The stock acid method.	(2.) The modified acid method.	(3.) The $Hg_2(NO_3)_2$ method.
	%WO_3.	%WO_3.	%WO_3.
(*a*)	71.3	72.4	72.6
(*b*)	68.8	69.6	69.8
(*c*)	67.3	67.1	67.4
(*d*)	68.2	—	68.1
(*e*)	59.2	—	59.5
(*f*)	71.3	—	72.4
(*g*)	—	71.5	72.4
(*h*)	64.6	65.3	—

It will be seen that method (1) tends to give the lowest results, which occasionally are below those of methods (2) and (3) by more than desirable limits, which may be placed at 0.6 % WO_3 on this class of material, e.g.—(*a*), (*b*), and (*f*). Method (2) agrees well with method (3), but the latter method inclines to give the higher results. It need only be stated, in conclusion, that the above results represent average results obtained in the writer's practice.

METHODS AND APPARATUS FOR THE DETERMINATION OF DUST SUSPENDED IN AIR.

By P. H. Warren and T. A. Read.

(*Read before the Broken Hill Branch*, 20*th June*, 1922).

Part I.

The quantity of dust present and the methods used for suppression of dust in metalliferous mines has, in recent years, received considerable attention from mining engineers in various parts of the world.

Shortly after the publication of the General Report of the Miners' Phthisis Prevention Committee, South Africa, in 1916, the officers of the Broken Hill South Limited started investigations into the conditions of the dustiness of the air in the underground workings.

As the methods used are not generally known in Australia, the writers are of the opinion that the present is an opportune time to place them before members of the Institute.

A large amount of the subject matter of this paper has been drawn from the 1916 and 1919 reports of the Miners' Phthisis Prevention Committee, South Africa, the paper of Mr. Innes published in "The Chemical, Metallurgical and Mining Society of South Africa"; also various articles appearing in the Journal of Industrial Hygiene, and the American Journal of Public Health.

A great deal of research work into the cause, prevention and estimation of mine dust has been made by the Miners' Phthisis Prevention Committee of South Africa, and officials of the United States Bureau of Mines. The writers considered it advisable to begin the investigation at the South Mine with apparatus and methods developed by the South African Committee.

Through the Australian agent of the Cape Explosives Company, inquiries were made preliminary to procuring the necessary apparatus for the determination of dust, from South Africa. The writers then learned that the older gravimetric method used by the Phthisis Prevention Committee was likely to be superseded by the konimeter method developed by Sir R. Kotze. Recognising that the konimeter instrument had some advantages over gravimetric methods, the Broken Hill South Limited purchased one of these instruments. This was the first Konimeter brought to Australia.

SIZE OF DUST PARTICLES IN THE LUNGS OF MINERS.

The following information is presented to give an idea of the size of particles that occur generally in mine dust.

In appendix No. 91 by Dr. Moir, published in the general report of the Miners' Phthisis Prevention Committee 1916, and reprinted in appendix No. 6 in the 1919 report, it is stated :—

> "The great majority of the quartz particles in such specimens of lung dust are less than 2* microns in diameter. None of them have been found to exceed 14 microns, and particles of such size as this are relatively rare, say 1 in 100,000. Very occasionally an acicular or elongated wedge-shaped particle of quartz is found."

A record of the dimensions and shapes of 120 particles of lung dust which have been separated, is as follows :—

Of the 120 particles, 117 were quartz, 2 rutile, and 1 probably tourmaline. The true ratio of the quartz to the more birefringent minerals was, however, most certainly much higher, probably 400 : 1.

The following is a summary of the "average diameter" of these 120 particles, the average diameter being the geometric mean of the length and breadth of each particle :—

15	Particles	less than	$\frac{1}{3}$	micron
16	,,	about	$\frac{1}{2}$	,,
12	,,	,,	$\frac{3}{4}$	,,
30	,,	,,	1	,,
15	,,	,,	1-$\frac{1}{3}$	,,
13	,,	,,	1-$\frac{2}{3}$	,,
11	,,	,,	2-$\frac{1}{4}$	,,
3	,,	,,	2-$\frac{3}{4}$	,,
3	,,	,,	3-$\frac{1}{2}$	,,
1	,,	,,	4	,,
1	,,	,,	5-$\frac{1}{2}$	,,

---

* 1 micron equals $\frac{1}{25000}$ part of an inch.

There appeared to be some selection of particles about 1 micron in average diameter. The mean of the average diameters of the 120 particles was 1.2 microns.

Dr. Watkins Pitchford in Appendix 8, 1916, report, of Miners' Phthisis Prevention Committee, remarks in regard to size of particle visible in lung section :—

> "With regard to the dimensions of the particles, it is to be remarked that the length of all of them (with the exception of the third examined) fell between 1.1 and 13.0 microns, and they are classified as follows :—
>
> Classification according to length—
>
> | | | | | | |
> |---|---|---|---|---|---|
> | 14 | particles between | 1 | and | 4 | microns |
> | 39 | ,, ,, | 4 | ,, | 6 | ,, |
> | 29 | ,, ,, | 6 | ,, | 8 | ,, |
> | 12 | ,, ,, | 8 | ,, | 10 | ,, |
> | 6 | ,, ,, | 10 | ,, | 13 | ,, |
>
> Mean dimension of 100 particles, 6.13 x 2.18 microns, (The third particle was omitted)."

In Bulletin No 132, U. S. Bureau of Mines—"Siliceous Dust in relation to Pulmonary Disease among Miners in the Joplin District"—by Higgins, Lanza, Laney and Rice—page 31 :—

> "In the course of the investigative work carried out by Dr. Lanza prior and subsequent to his preliminary report, many samples of sputum were collected from miners who reported for examination. These samples were centrifuged and small portions placed on glass slides to be examined under the microscope for tubercle bacilli. Siliceous dust in varying quantities was present on all of them; the cherty flint particles were plainly in evidence, and in some cases scattering particles of zinc blende and galena were recognised. Seven of the slides were examined by Dr. Thiessen, and the following is a portion of his report :—
>
> > "There are comparatively few particles of dust measuring more than 10 microns in diameter, but a large number measure between 5 and 2 microns with rapidly increasing numbers of smaller sizes down to

particles of colloidal dimensions. From this I should say we can take it that comparatively few particles of up to 10 microns ever reach the lungs.'

In the same Bulletin, suspension tests were made by exposing slides coated with cedar oil, to determine the size of dust particles suspended in mine air, and some of the figures obtained are given below. The temperature was 62°F, and the relative humidity ranged from 95 to 100%. Most of the time there was visible water-vapour in the air, which doubtless hastened the settling of the dust particles.

Slide No.	Mine.	Date.	Length of time exposed. In Minutes.	Cause of Dust.	Lapse of time between production and sampling of dust.	Size of largest particle of dust. Microns.
9	G	1915. May 1	10	Blasting	45 min.	10 x 10 Few particles larger than 5 m.
11	,,	,,	,,	,,	1 hr.	10 x 10
12	,,	,,	,,	,,	1 ,, 20 min.	10 x 8
14	,,	,,	,,	,,	1 ,, 40 ,,	10 x 5 Increasing number from 1 to 2 m.
16	,,	,,	,,	,,	2 ,,	10 x 5
17	,,	,,	,,	,,	2 ,, 30 ,,	9 x 6
18	,,	,,	,,	,,	3 ,,	5 x 5 Gradual decrease in size
20	,,	,,	,,	,,	3 ,, 40 ,,	Larger number of particles smaller
21	,,	,,	,,	,,	4 ,,	Size decreased and number increasing
22	,,	,,	,,	,,	4 ,, 30 ,,	Countless numbers of particles from 1 m. to sub-microscopic size

The following extract is taken from the first interim report of the Technical Commission of Inquiry to investigate the prevalence of miners' phthisis and pneumoconiosis at Broken Hill—page 75 :—

" *Table* 28.—Relative numbers of particles smaller and larger than 7.5 microns in maximum diameter present in the lung tissue of Broken Hill miners.

Note.—Samples of lung tissue were obtained post mortem and prepared for microscopic examination. Several sections were prepared from each lung from different portions which showed pathological changes.

Specimen.	Fields Counted.	Particles Counted.	
		Under 7.5 microns.	Over 7.5 microns.
1	5	5`1	1
2	3	379	3
3	4	327	1
4	4	472	0
5	3	398	2
6	3	335	0
7	4	323	5
8	4	776	1
		Total 3431	13

The above figures show that 3431 particles under 7.5 microns were counted for 13 over that size. If the number of smaller particles be multiplied by 400—the average ratio between the size of the fields and the areas of the sections—we see that for every particle of the dust in the lung having a greater diameter than 7.5 microns, at least 100,000 are present which have a smaller diameter.

No quantitative significance is attached to these figures; they merely show that the number of dust particles over 7.5 microns in diameter which accumulate in the lung is negligibly small compared with the numbers of particles smaller than 7.5 microns."

A study of the foregoing extracts serves to show that the greater proportion of dust that it is the aim to suppress is extremely small, and also leads to an arbitrary classification of the mine dust into—

Injurious dust particles 5 microns in diameter or under.

Non-injurious dust particles over 5 microns in diameter.

The foregoing investigations, with the exception of that conducted by the Broken Hill Technical Commission of Inquiry, deal with dust of a highly siliceous character. The dust occurring in the Broken Hill South Ltd. has a very different chemical composition, being formed from ores having the following analyses :—

	Siliceous.	Calcitic.
Insoluble	37.71	30.12
Lead	15.50	13.40
Copper	.10	.20
Arsenic	.18	.43
Antimony	.03	.06
Iron	4.90	3.03
Alumina	2.40	1.25
Manganese	2.47	1.70
Lime	4.00	15.70
Zinc	17.00	12.60
Sulphur	13.80	9.70
Carbon Dioxide ..	1.30	11.60
Fluorine	.24	.078
	99.63	99.868
Silver	11.7 ozs.	3.2 ozs.

The insoluble portion from the above has the following composition :—

	%	%
SiO_2	67.6	56.0
FeO	4.6	6.6
Al_2O_3	4.6	3.5
MnO	19.5	17.7
CaO	3.0	13.3
MgO	.9	2.5
	100.2	99.6

In addition to mining of sulphide ores, considerable development work is done in the country rocks surrounding the ore-bodies, which are chiefly schists of varying degrees of silicification.

The following analyses are of typical country rocks :—

Mineral.	Foot Wall Rock.	Hanging Wall Rock.
Water H^2O hygroscopic ..	.16	.36
Water H^2O combined ..	.97	.30
Silica SiO_2	60.34	67.09
Alumina Al_2O_3	19.10	14.57
Ferrous Oxide FeO	7.89	6.33
Ferric Oxide Fe_2O_3	.64	.17
Lime CaO	4.86	4.10
Magnesia MgO	.62	.96
Manganuous Oxide MgO ..	.93	.99
Soda Na_2O	.26	.35
Potash K_2O	3.83	4.03
Carbon Dioxide CO_2 ..	.04	.04
Titanium Dioxide TiO_2 ..	.93	.65
	100.57	99.94

As the dust resulting from work in these rocks and in the ore-bodies might be considered dangerous, both on account of its physical and toxic character, the importance of arriving at reliable data in regard to its quantity and chemical nature led to the employment, for its determination, of both the konimetric and gravimetric methods. The konimetric method was used for measuring the relative dustiness of the air in terms of numbers and sizes of particles present, while the gravimetric method was used for determining the weight of dust present in the mine air, and its chemical composition.

The fact that most of the dust particles found in deceased persons' lungs are very small, led to the development of the konimetric method of sampling in South Africa. The need was felt for some method which would give a measure of the number of dust particles existing in mine air.

A cubical particle having an edge of 20 microns has 8000 times the weight of a particle with an edge of 1 micron. It will therefore be seen that one large particle, too large perhaps to find entrance to a man's lung, would, if taken into a gravimetric sample, seriously affect the result. The air of a place may be heavily charged with myriads of particles of from one to two microns in size, yet the result of a gravimetric sample would be comparatively light.

Considerable attention has been given to the development of the konimeter as an instrument giving a measure in terms of numbers of particles per unit volume of air, and it would also make possible an examination of the physical character of the particles.

Much has been written in comparison of the merits of both methods. The writers feel that, whereas most reliance can be placed upon the results of gravimetric sampling properly conducted, this method in any new investigation should be supplemented by the use of the konimeter.

In the South Mine investigations they used the konimetric method of the Miners' Phthises Prevention Committee of South Africa, as described in their 1919 report, from which the following description is taken. :—

" *The Konimetric Method.*—The konimeter is shown in Plate I., figs. 1 to 6. It consists essentially of a valveless cylindrical suction pump having a cylinder capacity of 5 cc. The piston rod when depressed is held by a catch (*a*) but when released by a trigger (*b*) the brass spring (*c*) forces the piston (*d*) up to the end of its stroke, thereby sucking in 5 cc. of the dusty air through the channel (*e*) the small chamber or capsule (*f*) and the nozzle (*g*). The nozzle is tapered and smoothly bored and has a diameter of 0.6 mm. at the narrow end, and the current of air impinges on the glass slide (*h*) which is placed 0.5 mm. from the nozzle, and is coated with vaseline. The capsule and all connections between the outside air and the piston are made small so that the amount of dead air space is reduced to a minimum. The glass slide (*h*) which is made by cutting the standard English microscope slide (3″ x 1″) in half, is held over the capsule (*f*) by the spring (*j*) which, together with the rubber ring (*k*) secures an air-tight joint. This ring is made with a diameter of 16 mm., thus permitting the slide to be shifted for the purpose of taking a number of samples on the one slide without any of the samples coming into contact with the ring.

The slides are carried in the holder shown in Fig. 6, and a number of these can be accommodated in a small portable box, together with the konimeter. The glass slides are placed face inwards in the holder and held in position by a steel spring. The centres of the holders are recessed to prevent the dust sample being rubbed off, and also to make the glass slide fit closely and prevent contamination. The glass slide is coated with good grade veseline.

Before taking the sample, the piston is depressed and released a few times with a dummy slide on, so as to remove foreign dust. The tightness of all parts should also be tested by placing the finger over the outside end of the nozzle. On releasing the spring, the fact that the piston moves only slowly and then keeps almost still, will show that there is practically no leakage.

As the glass slide can be shifted over the capsule, and each sample is represented by a spot of dust, which is less than 1 mm. diameter, it is possible to take a number of samples on the one slide. A convenient number is six.

The strength of the piston spring should be sufficient to give a calculated velocity of air at the impinging nozzle end of not less than 30 nor more than 80 m. per second, unless the veseline coating is very thin, when the velocity can be increased to 130 m. per second. With a nozzle of 0.6 mm. in diameter, the stroke should take about one-third of a second.

The taking of a sample with the konimeter is almost momentary, and the small size of the instrument enables the operator to take the sample at any particular point required. As a rule it is necessary to ascertain the quality of the air as actually breathed by the workers, and the sample should therefore be taken in proximity to them and at the level of the mouth and nose. When the air contains but little dust, it may be necessary to repeat the stroke several times without removing the slide, so as to catch the dust from a larger volume of air than 5 cc. Similarly, for very dusty air, the stroke should be shortened, so as to take 1 cc. or 2 cc. of air only; this is done by the insertion of a stop over the spring.

The Microscope and the Method of Counting.—The stage of the microscope should be marked off so that spots may be readily found, or it is better still to have the instrument fitted with a mechanical stage.

The micrometer eyepiece may be ruled in several ways, some with sectors and some with squares. The sector ruling is the most convenient when the dust spot is light, while the square ruling is more certain for heavy spots. The sector ruling consists of two lines ruled at an angle of 9° or 18° to each other, with cross lines to facilitate counting. Two lines are ruled outside the sectors at such a distance that they appear to be 5 microns apart, as a guide to the eye to enable the size of the particles to be estimated. With heavy dust spots, *i.e.*, 500 particles per cc. or over, the counting with sector ruling is very tiresome to the eyes.

The square ruling consists of approximately 20 lines ruled at right angles, forming 400 squares. Two pairs of parallel lines are ruled through the centre of the diaphragm to enable an estimate of the plus and minus 5 microns sizes of dust to be made.

The most suitable magnification is about 180 diameters. The examination is best made with an objective of rather low power and an eyepiece of high power. While for certain purposes the definition is not so good as with the reverse arrangement, greater depth of focus and better contrast are secured. A suitable combination is a two-thirds inch objective and an ocular magnifying 10 to 12 times.

It is desirable to adjust the tube length of the microscope correctly. This is done by smearing thinly a little human blood over a clean slide and carefully focusing under the microscope ; the slide is moved around until a corpuscle is seen that looks like a circular disc, thus showing that it is lying flat on the glass. which is important. As human blood corpuscles are between 7.5 and 8 microns in diameter, we have a convenient standard for judging the width of the pair of fine parallel lines. The tube length of the instrument is so adjusted that the corpuscle is 1.5 times the space between the two lines.

Counting.—If the sector ruling is used, the following is the usual procedure :—The sample is focussed and adjusted so that its centre corresponds with the intersection of the sector lines, and the particles lying between the lines on both sides of the centre are counted. The number found is multiplied by 2 if the sector is 18°, and by 4 if it is 9°, in order to ascertain the number of particles per cc. The number of particles over 5 microns is similarly determined.

If the square ruling is used for counting, the sample is focussed and adjusted centrally over the spot. The number of squares covered by the whole sample estimated, the dust in a number of squares regarded to be average of the whole spot are counted and an average taken."

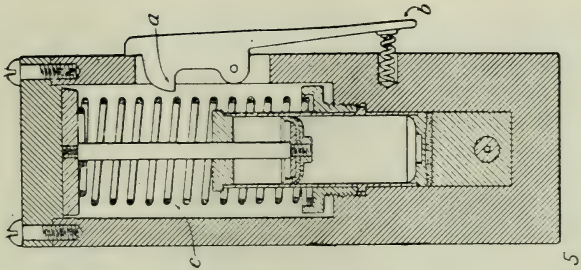


Fig 5 Section on AB

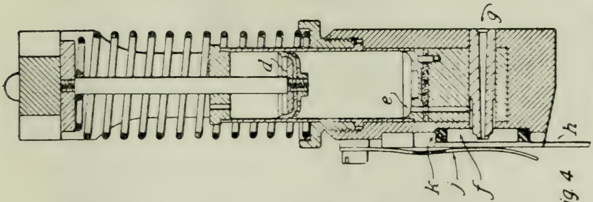


Fig. 4 Section on CD

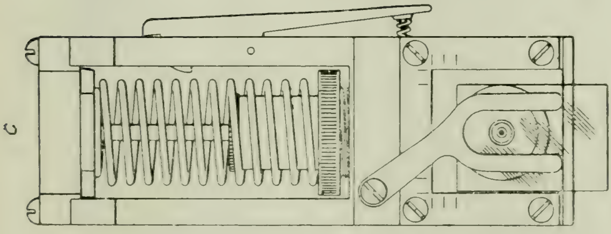


Fig. 2 Front elevation

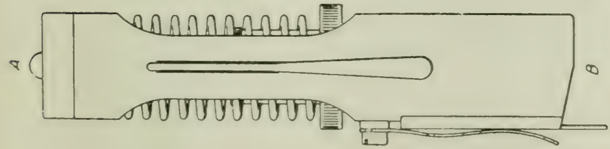
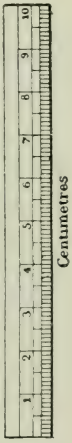


Fig. 1 Side elevation

THE KOTZÉ KONIMETER



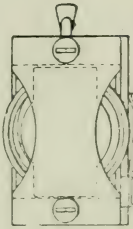


Fig. 3 End view

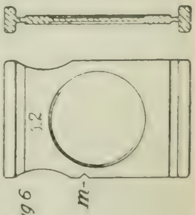


Fig 6

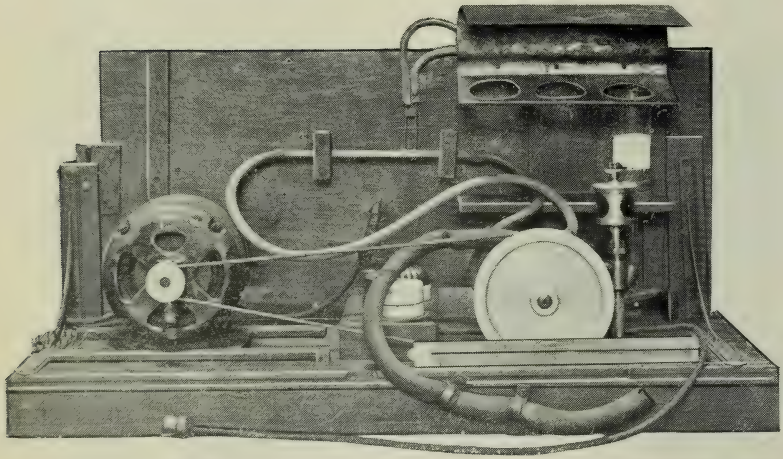


FIG. 1.
Crowell Pump Unit—Front view.

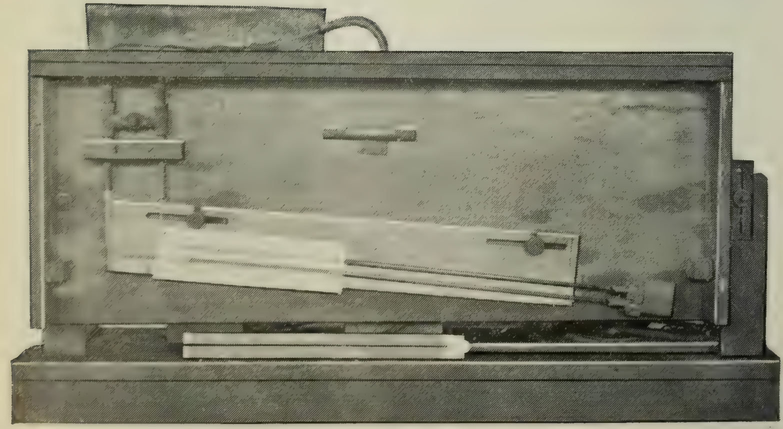
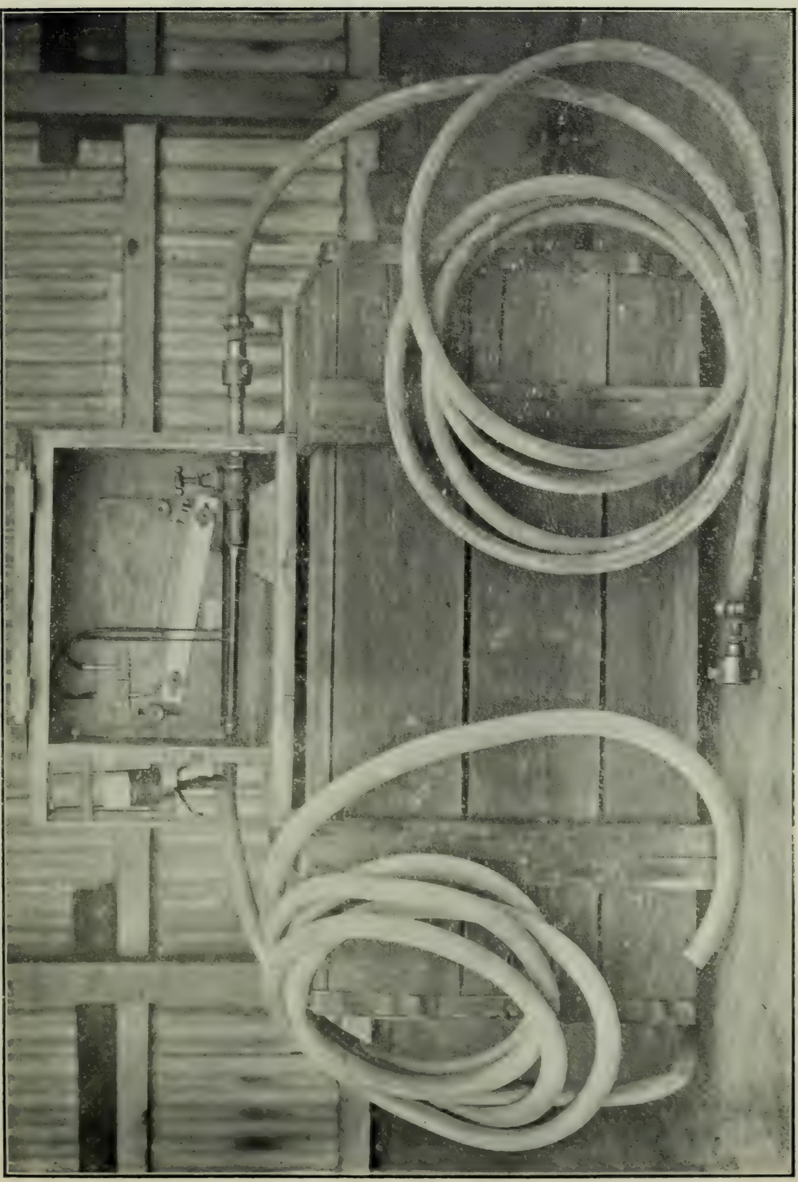


FIG. 2.
Crowell Pump Unit—Back view showing Manometer.



The Kotze Pump with hoses arranged for aspirating air through sugar tube.

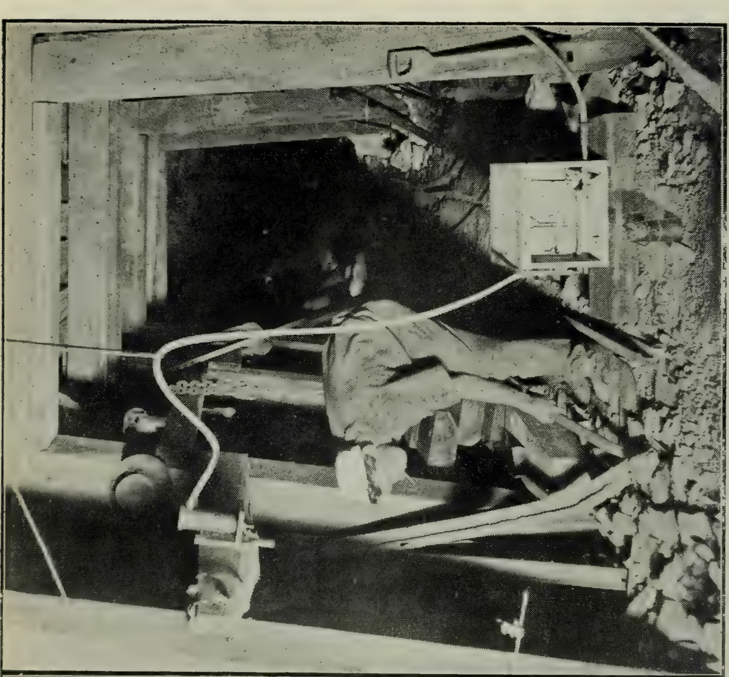


FIG. 1.
Shovelling in Gangway.

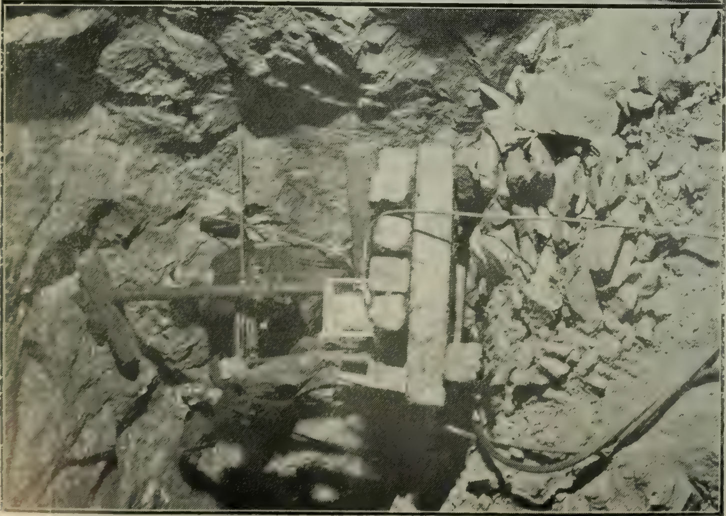


FIG. 2.
Boring in open Stope.

Securing dust samples with Kotze Pump and Read Water-spray Apparatus.

Notes on Konimeter.

South Mine experience with the original Kotze Konimeter showed the existence of one or two defects in construction which have been eradicated in the design shown in Figs. 2 and 2*a* by Mr. J. C. Cunningham, Chief Engineer on the Broken Hill South Ltd.

The defects found were :—

(1) Leakage of air under the glass slide owing to the uneven pressure exerted by the spring clip *j*, Fig. 4, plate 1.

(2) The possibility of leak through the joint between the body of the instrument and the piston cylinder.

These two defects were taken care of in the instrument now used, which is made of brass throughout, the cylinder being bored in the body of the instrument. Two side spring arms hold the glass in position and exert an even pressure thereon. The suction release spring travels in a brass cylinder and is attached to a disc, which ensures good contact with the cylinder walls. A bracket and centering device is incorporated in the design to permit of the use of a low-power microscope for examining dust spots in routine work.

Gravimetric Methods.

The work of securing gravimetric dust samples, in essentials consists of two operations except in the case of electrostatic precipitation—

(1) Measurement of the volume of air to be sampled.

(2) Filtering such measured volume of air through some collecting medium such as sugar or water.

The final result, usually rendered in terms of milligrams of dust per cubic metre of air, depends upon the care and skill expended upon the work. There are numerous factors in mines affecting the efficiency of the work, but one of the most important is that the equipment should be durable, light and portable, as samples require to be secured in places difficult of access, such as deep winzes and rises. The means available at present for securing gravimetric samples in a general way of speaking, are as follows :—

(1) THE ELECTROSTATIC METHOD OF DUST COLLECTION.

This method is described in the Journal of Industrial Hygiene, November 1919, in an article by J. Pentiado Bill, M.D.

Efficiencies up to 82.3% have resulted from the employment of this method in laboratory test, but its use underground in its present form could not be recommended at present, for the following reasons :—

1. It would be necessary to carry electrical wires to each place sampled.
2. Considerable electrical apparatus is necessary, thus departing from the ideal of cheapness and portability.
3. The very high voltage necessary—about 20,000—would be a source of danger, especially in confined spaces such as developmental deadends.
4. Working difficulties in the way of keeping the dust sample intact and uncontaminated till it reaches the assay office to be weighed, would be more pronounced than is the case with sugar tubes.

(2) SUGAR TUBE METHODS.

Sugar tube methods of collecting dust have been extensively used, and efficiencies running from 70% to 98% by the weight method, depending chiefly on the degree of fineness of the sugar used, have been obtained in laboratory experiments ("Determination of Rock Dust in Air," by Fieldner, Katz and Longfellow—Tech. Paper No. 278, U.S. Bureau of Mines, 1921). But by the tobacco smoke method, efficiencies only of 40%-60% have been obtained.

The method, in general, possesses the following advantages :—

1. The tubes are easily and quickly handled with relatively small danger of loss or contamination of the sample.
2. Three or four samples may be secured in the course of one morning's work, as there is very little delay in attaching fresh sugar tubes to the aspirating equipment.
3. The equipment necessary may be arranged in a light and portable unit.

The disadvantage of this method for collecting samples is that a contamination already exists in the sugar in the form of dust and is often as great as the dust collected from the air.

The chief difficulties met with are those common to all methods of securing gravimetric samples in places difficult of access, and centre round the application of the force necessary to aspirate the dusty air through the sugar. This may be done in several ways, viz. :—

(A) HAND PUMP.

This method is a standard method in Johannesburg, having been adopted as the official apparatus in 1913. It is an adaptation of the ordinary motor tyre pump, so constructed as to be double acting and to utilise only the inlet, the air passing through going to waste. The pump, as used in South Africa, is illustrated in the General Report of the Miners' Phthisis Prevention Committee, on page 73, and also in the final report on page 8.

A lungmotor pump, hand operated, has been used on the Broken Hill South Ltd. for obtaining samples in this way.

Accurate work can be done with the hand pump, but this method of aspirating air is subject to criticism for the following reasons :—

1. The measurement of the volume of air sampled depends on the correct count of the number of strokes being taken. The mechanical counter provided for this purpose is quite satisfactory, but reliance on the personal factor would tend towards error.
2. In order to secure efficiency with the ordinary type of sugar tube, the rate of aspiration should not be faster than one cubic foot of air per minute. The total sample should be at least 30 cub. ft., *i.e.*, .85 cub. m. which would mean 600 strokes of the pump. Where the air is known to be comparatively free from dust, the volume sampled should be much greater in order to secure a sufficient weight of dust to be accurately handled in the assay office, and also to minimise the effect of insol. dust contained as impurity in the sugar. Suppose, for example, a sample has been taken for 30 minutes at 32.5 litres per minute ; weight of sugar equals 50 grm. ; weight of impurity in sugar equals 2.6 mg. Assume air to contain 1 mg. of dust per cub. m. of air, the total weight of dust will be 3.6 mg., 72% of which consists of impurity in the sugar. It is therefore advisable to sample in such a place for at least 60 minutes in order to increase the percentage of dust in the air to the total weight of dust in the sugar. The number of strokes per sample then becomes about 1200.

3. Each stroke of the pump should be the full length and the speed should be kept regular.
4. It is essential that the pump be frequently calibrated against a meter, for capacity, as there is a tendency towards leakage at the valves owing to the pressure drop in the circuit.
5. The tendency is towards the use of pulverised sugar, imposing a resistance of approximately 40 in. of water upon the circuit. Not only will the tendency to leakage mentioned in No. 3 become more pronounced under these conditions, but greater physical effort to accomplish the work of pumping will be called for.
6. From the foregoing it is claimed that although it is possible to obtain reasonably accurate results by means of the hand pump, continuous routine work would impose too great a strain on the operator, and that an average intelligent white man would refuse to remain long associated with such a treadmill occupation, and that in any case the personal factor bulks too largely in the operation.
7. Finally, it should be pointed out that this method is unsuitable for obtaining samples in very dusty atmospheres such as are found immediately after firing, as the operator is exposed to dust during the whole time of sampling.

(B) THE CROWELL PUMP.

This small rotary pump is light, and may be adapted for numerous purposes. Dr. S. A. Smith of the Broken Hill Technical Commission, who had employed a Crowell pump for administering anaesthetics, suggested its use for aspirating the air in securing gravimetric samples.

The apparatus as formerly used at the South Mine, and subsequently by the Technical Commission, is shown in Figs. 1 and 4, and Plate II. and consists of—

A Crowell pump belt driven by a $\frac{1}{4}$ h.p. motor at 700 r.p.m. The suction end of the pump is connected by $\frac{3}{4}$-in. rubber and brass tubes to a four-way distributor. Three of the outlets serve to connect by rubber tubing to the sugar tubes. Three tubes may be used at one time, but many samples have been collected using one only. A pair of pitot tubes placed between the sugar tubes and the pump suction, measure

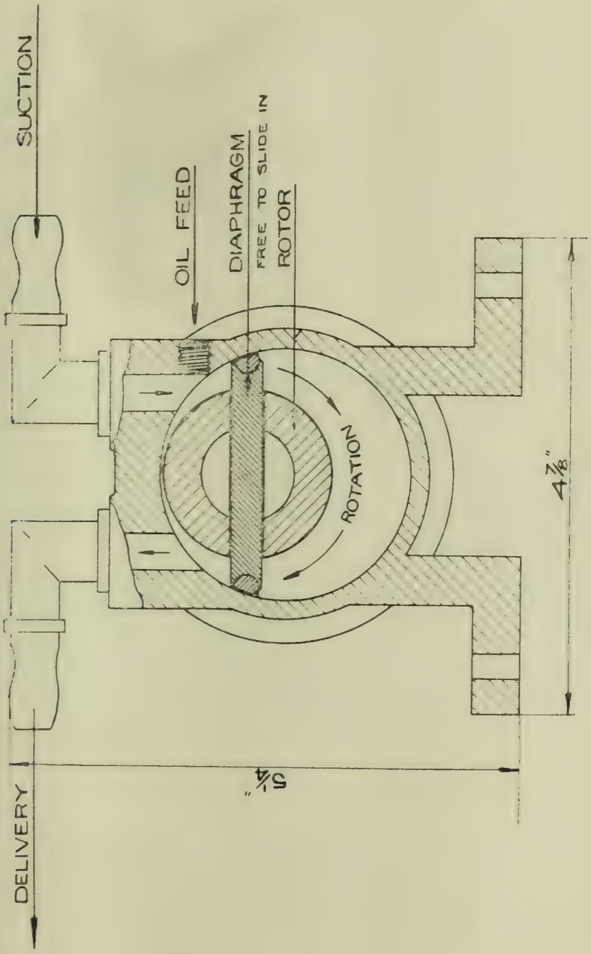
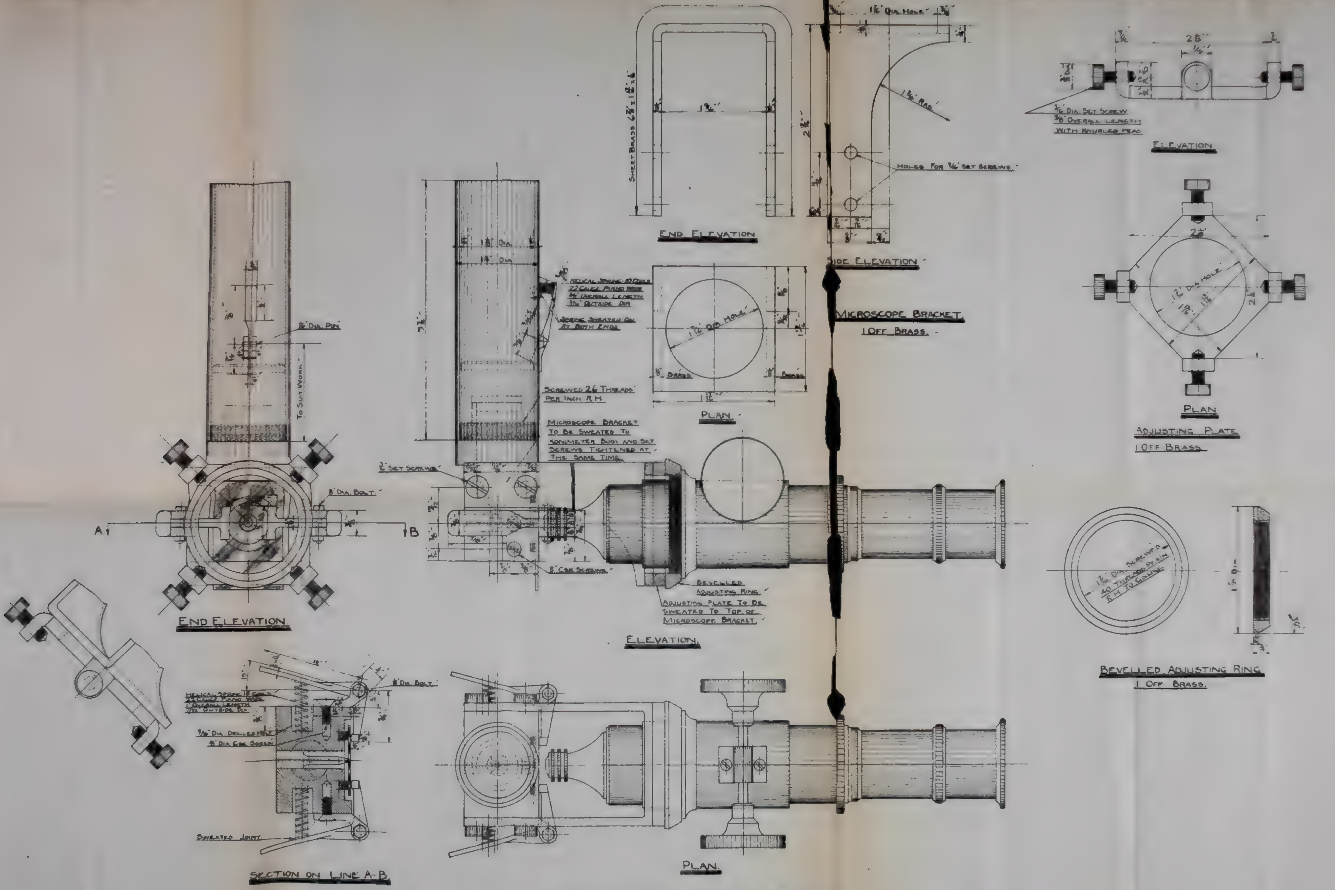


Fig. 1.
Crowell Pump.

the volume of air aspirated. The pitot tubes are calibrated by direct aspiration of air into a closed air-tight tank. The pump aspirates from 50 to 60 litres per minute. The sugar tubes are cut from cylindrical glass tubing. The bottom of the tube is closed with a good-grade single-hole rubber stopper. Through the hole passes a ⅜-in. diam. glass tube to connect with the pump. In the bottom of the tube is placed a wire-gauze filter, raised from the rubber stopper with a glass ring ½-in. high, to allow of free air circulation. The sugar for filtering the air and collecting the dust is placed on the upper surface of the wire gauze, 50 grm. being the usual weight of sugar used in each tube. The sugar in the tube is protected from extraneous dust by a tightly fitting rubber stopper. The sugar used is a good-grade coarse sugar screened on 20 and 80 mesh commercial screens ; the product minus 20 plus 80 is used for the sugar tubes. A suitable case is provided for carrying the sugar tubes and to prevent breakage. The whole apparatus is so mounted that two men can carry it comfortably to any locality in the mine.

The method of collecting the sample is quite simple, viz.—The apparatus is taken to the stope or other working place, and set up over a couple of 10 in. x 10 in. timbers or other level surface. The wiring to the motor is connected and the outfit given a preliminary run. The three tubes, each containing 50 grm. of clean sugar, are connected to their respective connections and the motor started. The apparatus is allowed to run for a sufficient length of time to give a sufficiently large sample to work with. During the period of sampling, the pitot tube reading is taken periodically, the apparatus always being levelled and the guage readings brought back to a zero mark. The time of running must also be noted to calculate the volume of air sampled. Having taken the samples, the motor is stopped, the sugar tubes closed at the top with a rubber stopper and then removed from the stand to the case for carrying them. The samples are then taken to the chemical laboratory for examination.

The Crowell pump has proved quite satisfactory for aspirating the air, and can be used with either electrical power or compressed air as a motive force. Used with electrical power, it is, of course, necessary to wire each place in advance of sampling operation. This feature limits the use of the equipment as described to those mines which have electrical power underground. It is also necessary to supply a motor suitable for the periodicity and voltage of the lighting circuit.



MODIFIED KOTZE KONIMETER WITH MICROSCOPE ATTACHED.
SCALE FULL SIZE.

Development of Power Unit.—As some mines have no lighting circuits below the surface, attention has been given to the possibilities in the way of providing a compressed-air prime mover. Two Crowell pumps were set up on the same base and their central shafts mechanically connected ; compressed air was then introduced to the inlet of the

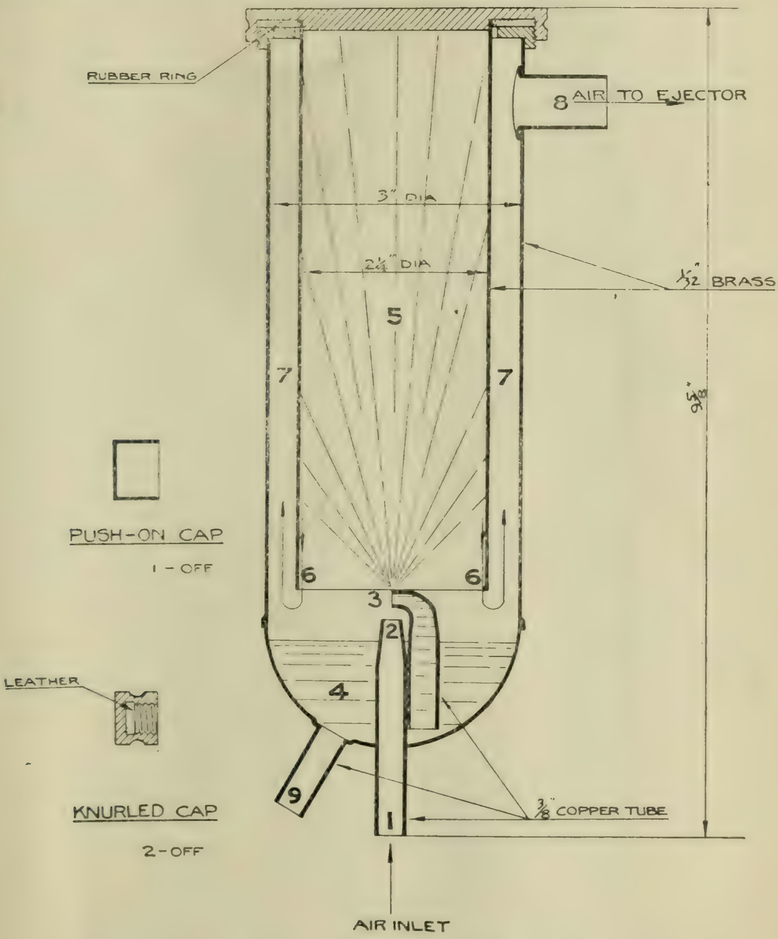


FIG. 3.

WATER-SPRAY AIR SAMPLER.

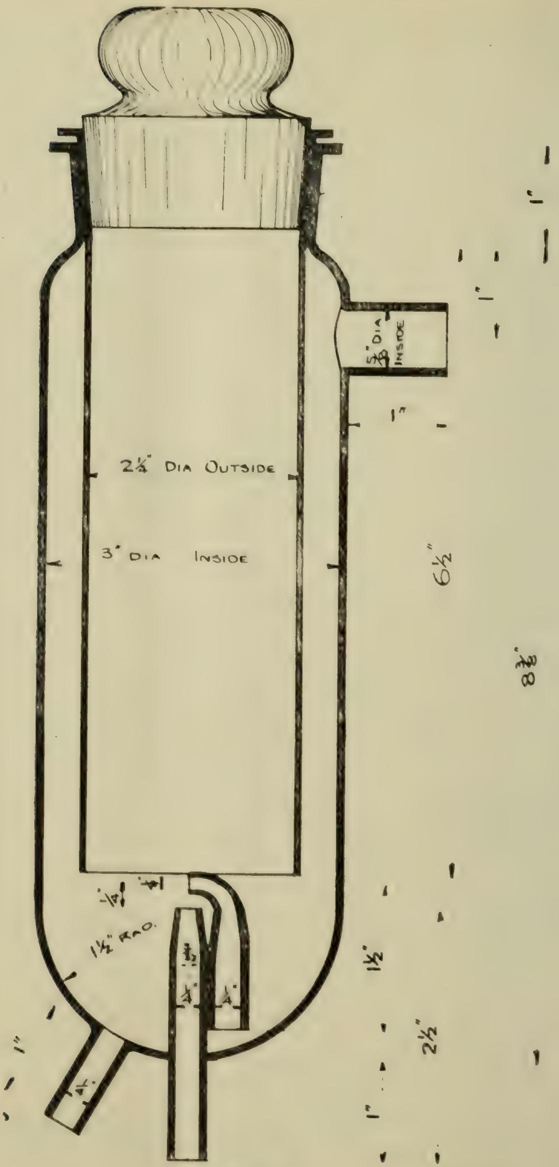
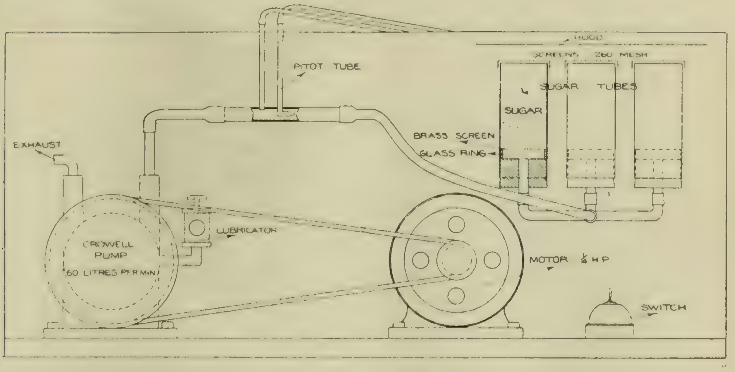


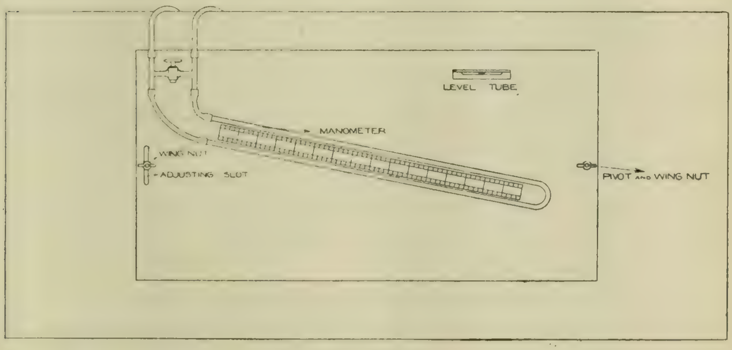
Fig. 3a.

Glass Water-spray Air Sampler. (Half Scale) motor pump, thus causing the fan shaft of the other to aspirate air in the usual way. This scheme is quite successful, but is subject to the following criticisms :—

1. The unit costs about £30.



FRONT ELEVATION



BACK ELEVATION

Fig. 4

Sketch showing Crowell Pump Unit for aspirating air through Sugar Tubes.

2. Crowell pumps, especially when their efficiencies drop from wear, require a tremendous quantity of oil relative to their small size.

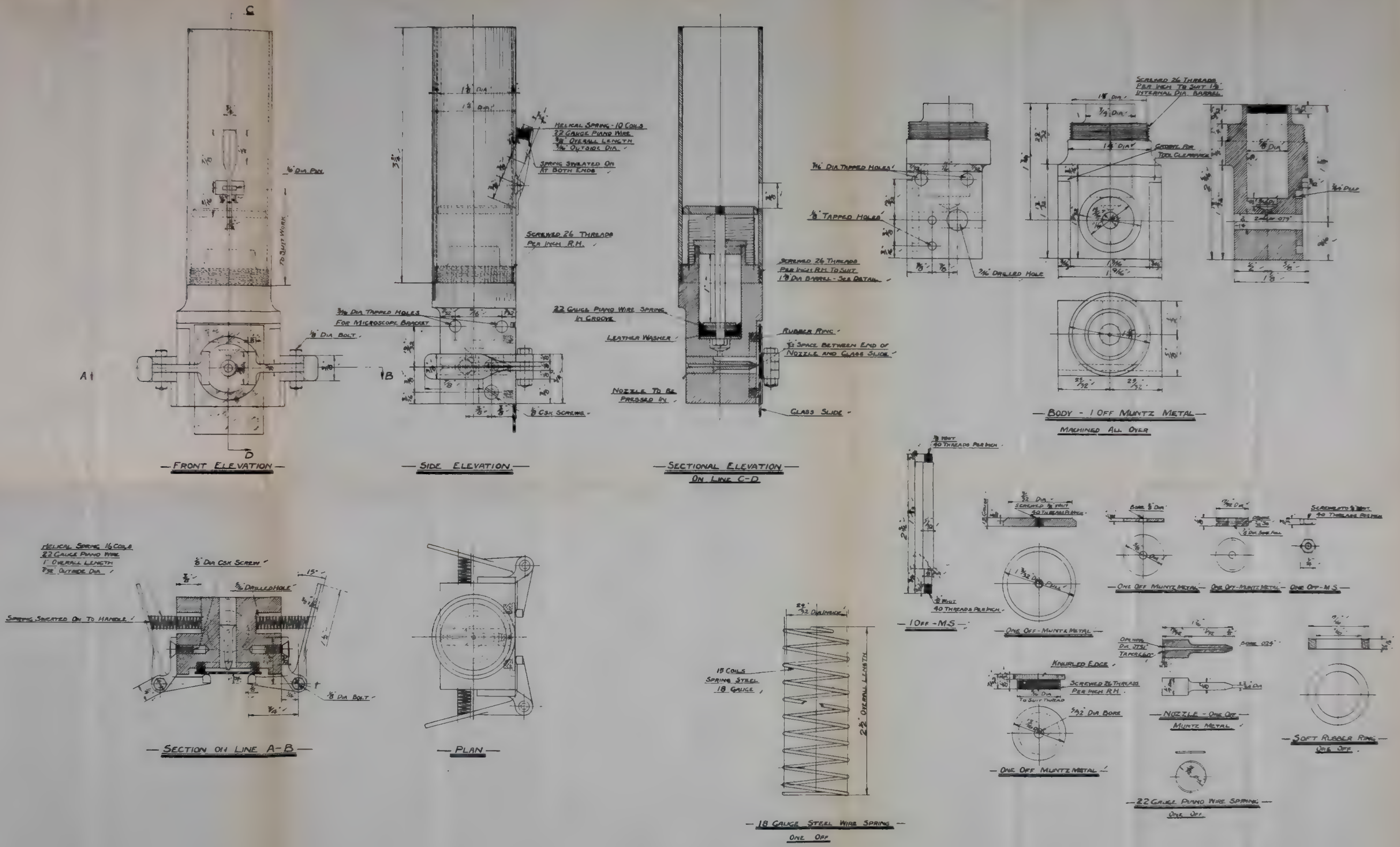
A "Little Giant" air-operated wood-boring machine was also fitted up to turn a Crowell pump. This scheme also was quite successful, subject to the same criticisms as in the case of the two connected Crowells. The capital cost is somewhat heavier—about £36.

There appears to be no reason why a cheap and light air-operated motor should not be designed to drive a Crowell for this purpose.

(c) The Kotze Pump or Ejector.

This simple, cheap and effective appliance, originally designed by Sir Robert Kotze of Johannesburg, has been incorporated into a very suitable unit for securing gravimetric samples. It combines the qualities of lightness, portability and durability, as shown in the accompanying illustration, Fig. 5 and Plates III. and IV. The volume of air sampled is measured by means of a pitot tube and scale calibrated against a Glover gas meter adjusted for 10 in. of water guage. The appliance possesses the following advantages :—

1. *Lightness and portability.*—The box without hoses weighs about 12 lb. It is necessary to carry about 50 ft. of hose in order to effect connection with the nearest air main. 6-ply hose $\frac{3}{8}$ in. diam. internal, is suitable. A special tee with a blind spud to close the end of the air main is also required, in case the machine hose be disconnected.
2. *Cheapness.*—The cost of the whole equipment, including hoses, is not more than £5.
3. Tests in the Broken Hill South Ltd. showed the appliance to be perfectly regular in working, more so indeed than the Crowell pump.
4. Samples may be taken in any place where compressed air is provided, and no preliminary arrangements are necessary.
5. Absence of oil makes for clean work.
6. The ejector is easily capable of dealing with 100 litres of air per minute (about $3\frac{1}{2}$ cub. ft.) but as it is advisable for efficient work in sugar-tube sampling to limit the quantity to 32.5 litres per minute, the quantity of compressed air used is so small as to be incapable of measurement on a Clark air meter. The minimum capacity of which at 95 lb. pressure is about 7 cub. ft. of free air per min.


C
D
A
B
1/16" Dia. Pin
To Suit Work
— Front Elevation —
1 1/8" Dia.
Helical Spring - 10 Coils
22 Gauge Piano Wire
3/8" Overall Length
3/16" Outside Dia.
Spring Sweated On At Both Ends
Screwed 26 Threads Per Inch R.H.
3/32" Dia. Tapped Holes For Microscope Bracket
1/8" Dia. Bolt
1/8" Csk Screws
— Side Elevation —
22 Gauge Piano Wire Spring In Groove
Leather Washer
Nozzle To Be Pressed In
Rubber Ring
1/32" Space Between End of Nozzle and Glass Slide
Glass Slide
Screwed 26 Threads Per Inch R.H. To Suit 1 1/8" Dia. Barrel - See Detail
— Sectional Elevation On Line C-D —
3/16" Dia. Tapped Holes
1/8" Tapped Holes
3/16" Drilled Hole
Screwed 26 Threads Per Inch To Suit 1 1/8" Internal Dia. Barrel
1 3/8" Dia.
3/4" Dia.
1 1/4" Dia.
Groove For Tool Clearance
5/8" Dia.
3/16" Deep
1 3/16"
1 9/16"
1 1/8"
— Body - 1 Off Muntz Metal —
Machined All Over
1/8" Whit. 40 Threads Per Inch
1/8" Whit. 40 Threads Per Inch
— 1 Off - M.S. —
Screwed 1/8" Whit. 40 Threads Per Inch
1 3/32" Dia.
— One Off - Muntz Metal —
Bore 1/8" Dia.
— One Off Muntz Metal —
7/32" Dia.
1/8" Dia. Bore Full
— One Off - Muntz Metal —
Screwed to 1/8" Whit. 40 Threads Per Inch
— One Off - M.S. —
Knurled Edge
Screwed 26 Threads Per Inch R.H.
To Suit Thread
5/32" Dia. Bore
— One Off Muntz Metal —
Opening Dia. .0751"
Taper 60°
Bore .024"
— Nozzle - One Off — Muntz Metal
— Soft Rubber Ring — One Off
— 22 Gauge Piano Wire Spring — One Off
Helical Spring 16 Coils
22 Gauge Piano Wire
1" Overall Length
7/32" Outside Dia.
Spring Sweated On To Handle
1/8" Dia. Csk Screw
3/16" Drilled Hole
15°
1/8" Dia. Bolt
— Section On Line A-B —
— Plan —
29/32" Dia. Inside
15 Coils Spring Steel 18 Gauge
2 1/4" Overall Length
— 18 Gauge Steel Wire Spring — One Off

MODIFIED KOTZE KONIMETER
Scale—Full Size.

7. It can be used on any form of known dust-collecting apparatus.

It might be argued that the exhaust, partly consisting of compressed air, will vitiate the sample. This difficulty is not at all serious. In the first place, even when sufficient compressed air is being used to induce a flow of 100 litres of air per minute through the filtering medium, the quantity so used is negligible by comparison with the exhaust from a rock drill when boring; and secondly, a length of rubber tubing one inch in diameter can be attached to the exhaust pipe, as shown in the photograph, of sufficient length to carry the exhaust away from the place being sampled.

Choice of Method.

After full consideration of the whole problem, the writers have formed the opinion that gravimetric sampling of dust in mines may be successfully carried out by either the sugar-tube method or the distilled water method as used in the Read water-spray sampler, and that the latter method, for reasons set out later, is the better.

The equipment as used by the Broken Hill South Ltd. for sugar-tube sampling comprises—

1. A set of sugar tubes of uniform dimensions as shown in Fig. 6.
2. A Kotze air ejector mounted in a suitable case, with pitot tubes for measuring the quantity of air sampled.

Comparable results can only be obtained by standardising conditions as far as possible.

In the matter of collecting samples by this method, the writers think it advisable at present to follow the recommendations of Fieldner, Katz and Longfellow as detailed in report Tech. Paper No. 278, 1921, U. S. Bureau of Mines. Briefly, those recommendations are concerned with the following factors :—

1. *Size and shape of sugar tubes.*—The tube should be $2\frac{3}{8}$ in. diam. x $3\frac{1}{2}$ in. long in its widest part. The bottom end should be drawn down to a tube about $\frac{3}{4}$ in. diam., so that there is no sharp change in section. The object in this design is to ensure good contact of the dusty air with the sugar.

2. *Material used for tube.*—The tubes are commonly made from glass, but as those tubes are somewhat fragile, brass tubes of similar design to those recommended by the U. S. Bureau of Mines are being used. These tubes are illustrated in Fig. 6. The sugar rests upon a fine-mesh screen at the bottom of the cylindrical portion of the tube, and the tubes are fitted with a fine-mesh screen cap at the top to prevent the inclusion of accidental particles in the sample.

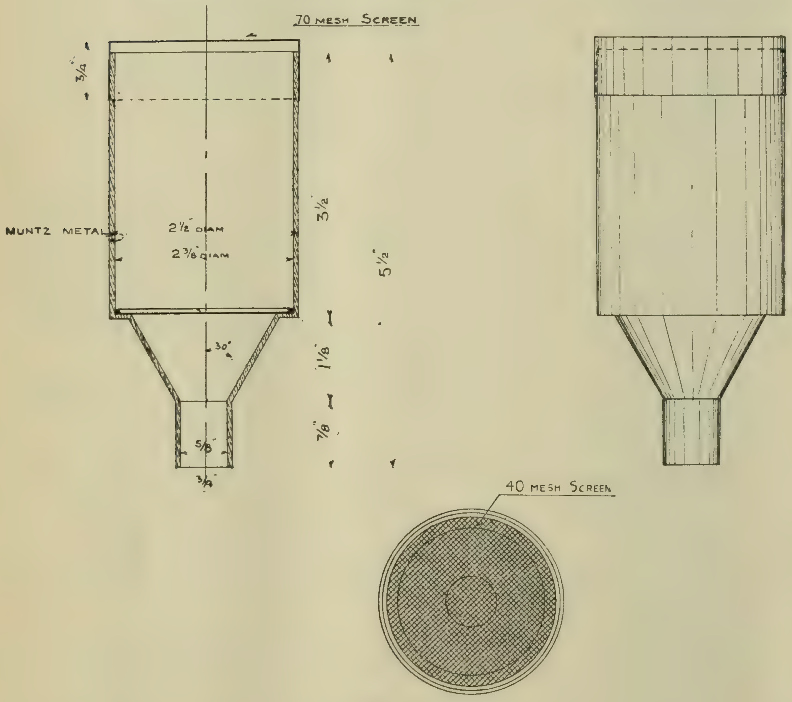


FIG. 6.
SUGAR TUBE FOR USE WITH GRAVIMETRIC DUST SAMPLER.

3. *Quantity and quality of sugar.*—The quantity of sugar used should not be so great as to impose a heavy resistance on the passage of the air through the circuit, but sufficient surface should be provided for dust deposition; 50 grm. is a suitable amount. Numerous experiments have been

conducted by the U. S. Bureau of Mines concerning the best size of sugar to use. Efficiencies from 70% to 90% have been obtained using granular sugar through 14 and on 65 mesh ; this is the size commonly in use in South Africa and locally. Pulverised sugar through 45 and on 150 mesh packed in the tube has yielded efficiencies up to 98% determined gravimetrically, but this practice brings the resistance of the circuit up to 40 or 50 in. of water. Ground sugar has not been tried out under working conditions, and it may be anticipated that in humid places it would prove a failure owing to coagulation. It therefore seems advisable to continue using granular sugar at present.

4. *Rate of flow of air.*—The velocity of the air through the sugar should not be too great, otherwise the finer particles of dust will escape the filter. With the size of tube and quantity of sugar above recommended, 32.5 litres per minute is a suitable volume.

The writers have adopted the practice of supporting the tube in a small wooden holder separate from the ejector box, to which it is connected by a length of rubber tubing. Samples may thus be collected near the nose and mouth of the men, and there is practically no danger of vitiation of air from the ejector exhaust. Similar technique is followed when using the Read water-spray sampler.

The distilled water method as applied in the Read appliance, illustrated in Fig. 3, makes use of the same general technique, but has the advantage of greater simplicity, and several variable factors are cut out. The quantity of water has not to be accurately measured, it being necessary only to see that the water remains at the level shown in the illustration, Fig. 3. There is no necessity to run blanks to determine insoluble impurity in the filtering medium. Danger of accidental dust is lessened as the tubes are covered at the top and the air sample is drawn upwards from the bottom. The equipment necessary possesses the same measure of portability and adaptability for general use as the sugar-tube method, the only difference being that the water-spray tubes are slightly larger. The Kotze ejector for aspirating and the wooden holder are used, as described in connection with sugar-tube sampling. The appliance has a high efficiency as shown by the tests recorded in Part II.

THE DETERMINATION OF THE DUST COLLECTED BY THE SUGAR.

The sugar is emptied out of the tube into a 500 cc. beaker, and the tube thoroughly washed out into the beaker with distilled water. Approximately 400 cc. of water is added to the sugar in the beaker, and the latter dissolved by stirring with a glass rod. The sugar solution is then filtered through a weighed Gooch crucible fitted with a filter of asbestos. The crucible, previous to weighing, is dried for two hours in the water bath. The crucible and contents are washed four times with cold distilled water and then dried in the water bath for two hours, cooled, and weighed. The increase in weight of the crucible represents the weight of dust contained in and collected by the sugar.

A blank test for weight of residue is always made on an equal weight of sugar, and allowed for.

The above procedure of weighing the sample is necessary in the case of dust obtained from the Broken Hill South owing to the high sulphide content of the dust.

The method of igniting the dust as adopted by the South African Committee cannot be used on the South Mine dust when exact figures are required; however, when comparative figures are required, the ignition method is used, in which case the sugar solution is filtered through a good grade filter paper having a low ash, and the filter paper and residue ignited in a porcelain crucible, and weighed.

THE DETERMINATION OF THE DUST COLLECTED BY THE WATER-SPRAY.

The dust-laden water is transferred to a 500 cc. beaker through the opening provided in the bottom of the apparatus, and the apparatus thoroughly washed out. The water is then filtered as in the above sugar solution. If a chemical examination of the dust is required, the water carrying the dust in suspension is evaporated to dryness on the water bath in a weighed platinum dish, cooled, and weighed. After weighing, the residue in the dish is treated by suitable methods for the determination of its constituents.

(3) WATER-SPRAY SAMPLERS.

The idea of passing dusty air through distilled water is attractive, as the filtering medium should be quite free from impurity, which is not the case with sugar. The water-spray principle has been made use of in the Palmer apparatus described by G. T. Palmer in the American Journal of Public Health 1916, 6-54.

Palmer, Coleman and Ward, in an article contributed to the above Journal, October 1916, also give descriptions of some test work on the Palmer and other apparatus. They claim very high efficiencies for the former.

Researches by Fieldner, Katz and Longfellow detailed in the Journal of Industrial Hygiene, September 1920, showed an efficiency of only 13% when the apparatus was used for collecting tobacco smoke from air, the catch being measured by a method based on light reflection. It is doubtful whether it is fair to test the efficiency of a water-spray apparatus in this way. Tobacco smoke particles, though uniform in size, are very small, averaging only .273 micron in diameter, and are not therefore representative of the size of dust particles ordinarily met with in industrial work. Further, they have an oily surface and are not therefore readily wetted by water. Possibly, higher efficiencies would have been obtained by the use of a light oil in place of distilled water as a collecting medium.

Tests conducted on a weight efficiency basis, using a weighed quantity of prepared silica dust, gave a return of 45%.

Smythe and Izard contributing to the above-mentioned Journal in September 1921, say, in criticism of the tests of Katz and his co-workers :—

> " The Tyndall (light reflection) phenomenon, depending largely on the presence of ultra-microscopic particles in the air, although the best absolute test of complete removal of particulate matter, is too severe a test of the practical hygenic efficiency of a dust sampling apparatus. Efficiency tests for such apparatus should be made with dusts containing particles at least as large as 10 microns in diameter, and the most frequent size in such dust tests should be one micron in diameter."

They further claim that had Katz used the original design of Palmer apparatus with a capacity of 5 cub. ft. per minute instead of the later 4 cub. ft. machine, his efficiency figure would have been considerably higher.

This apparatus, however, possesses an undoubted advantage in that the filtering medium is free from impurity.

Apart from other considerations, the Palmer apparatus in its present form is open to several objections for use underground:—

(1) The equipment is designed to operate with a small electrical motor, which method of aspirating the air imposes limitations upon its field of usefulness.

(2) A sampling equipment should permit of 3 or 4 samples being taken in one campaign. Palmer bulbs do not lend themselves to this practice, being both bulky and fragile.

The writers have for some time been working on a water-spray apparatus, designed by T. A. Read, in which the above-mentioned objections are overcome. This apparatus is shown in Fig. 3, and is the outcome of experiments conducted with an apparatus designed and described by V. G. Anderson in the Proceedings of the Society of Chemical Industry, Victoria, 1914. They found that in the Anderson apparatus, which was designed for gas absorption, the finely atomised water-carrying dust was drawn over, resulting in low and variable recoveries. The inner scrubbing chamber was therefore introduced, which completely retarded the water mist and dust.

The Read water-spray dust-sampling apparatus is based on the principle of dust-laden air entering the apparatus through the nozzle (1 and 2) at a high velocity. The difference of pressure provided by the power unit (a calibrated Kotze air ejector pump is used) causes a stream of distilled water to rise from the reservoir (4) at the bottom of the apparatus through the small nozzle (3), which is set at right angles to the air nozzle (2). The velocity pressure of the air is such that the water stream is atomised, with the result that the dust-laden air receives a thorough scrubbing in the inner chamber (5). The action of the water is such that the dust particles are removed from the air, and the latter, minus its dust content, passes beneath the baffle (6) up through the annular ring (7) and through the outlet pipe (8) whence it passes to waste through the ejector pump, its volume being measured en route by means of a pitot tube calibrated in litses per minute. The water containing the dust is removed through the drain pipe (9) and the interior of the whole apparatus washed out with distilled water.

The apparatus as used by the writers was made of sheet brass. Ordinary lead-tin solder should not be used to close the joints, as in the prerence of water electrolytic action sets up decomposing the solder and vitiating the sample. Either the joints should be brazed or a silver solder should be used.

The writers are having a sampler made in glass to the design shown in Fig. 3A which should obviate the difficulty referred to.

The baffle plate (6) plays an important part in the efficiency of the operation, as the atomised water is thus prevented from being drawn over with the air.

The dimensions of the sampler as used are given in Fig. 3. It is necessary that the outlet pipe (8) should be of large diameter in order to slow down the velocity of air passing through.

Highly satisfactory results, which are given in Part II. of this paper, have been obtained in test work.

The volume of water used in the apparatus is approximately 70 cc.

The capacity of the above-dimensioned apparatus is 45 litres air per minute.

The loss of water due to absorption by air is about .003 cc. per min., varying slightly with atmospheric conditions.

METHODS FOR CALIBRATING EJECTOR PUMP PITOT TUBES.

The general arrangement of pitot tubes and the ejector are shown in Fig. 5, Part I. The measurements given for diameters of the metal tubes are external, and the glass tubing internal. The glass gauge tubes are parallel, and set at an angle of approximately 10° to the horizontal. Since the drawing was made the arrangement has been slightly altered to permit of the Read water-spray sampler being used with the ejector as an interchangeable unit with the sugar tube. The new arrangement is shown in Plate IV., fig. 1. The dimensions of the ejector and pitot tubes remain unaltered.

The theory upon which the action of the guage is based is fully described in the first interim report of the Broken Hill Technical Commission of Inquiry, page 65.

Velocity of moving gas$=$v

m$=$mass of unit volume.

Energy given up per second$=\frac{1}{2}$ mv.

This change in momentum sets up a pressure which is balanced by a change in the level of the liquid in the gauge. The velocity pressure is delivered to the lower water meniscus by means of the dynamical tube. The absolute pressure of the gas is delivered to the upper water meniscus by means of the static tube, which reflects variations due to alterations in the resistance of the circuit and barometric changes. Thus in the lower tube we have static plus velocity pressure, and in the upper tube static only, as the opening in this tube is at right angles to the direction of flow of the gas. The movement of the liquid in the glass gauge tube therefore gives a measure of the difference of pressure.

In accordance with one of the well-known laws of friction, viz., $p=\frac{ks\ V^2}{a}$ the pressure varies as the velocity squared. The difference of pressure is balanced by a movement of the liquid giving a head H, Fig. 7, the pressure of which is Hgd ; d equals density of liquid, g equals gravitational constant. We use weak lime water, to which a few drops of phenol-phthalien have been added, producing a bright red color. d for this liquid is taken as 1. The use of lighter liquids increase the reading in inverse proportion to the density.

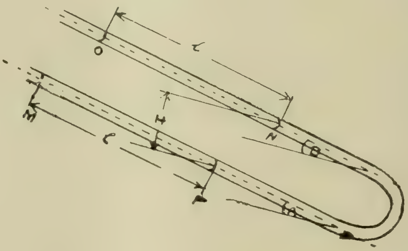


Fig. 7.
Pitot Tube

The tubes are of the same internal diameter ; therefore for a given change of pressure the distances moved by the two menisci will be equal, so that the head H which balances this change in pressure is caused by movement of the liquid from M to P and N to O. The component for the lower tube is 1 sin A and for the upper tube it is 1 sin B ; therefore H=1 (Sin A plus Sin B) ; or for a movement of 1 inch, H + Sin A plus Sin B.

In the above-mentioned report a curve has been designed which is correct for $\frac{1}{8}$-in. glass tubing. Should tubing of different diameter be used the curve would vary in accordance with the terms of the formula $p=\frac{ks\ V^2}{a}$

p=pressure in lbs. per sq. ft.
k=frictional constant.

s=rubbing surface which for constant length would vary with the internal diameter of the tube.
V=velocity of air.
a=area of section.

In the case under consideration, for tubes of unit length but different diameters, $\frac{s}{a}$ is the variable.

In the practical application of this theory the Crowell pump was set up in circuit with a large 20-light gas meter having a capacity of 2 cub. ft. per min. and adjusted for accuracy at a resistance of 10 in. of water column. The air, after having been drawn through suger tubes by the Crowell, was delivered to the meter from the outlet of which it passed to the pitot tubes of the ejector and so to waste. A water gauge was also included in the circuit in order to measure the resistance for each test. The pump was started and the time taken for the meter to register 5 cub. ft. noted by stop watch. The water-gauge reading was also noted. The point reached by the lower water meniscus in the glass gauge tubes was marked on the scale. The quantity passed through the meter was calculated in terms of litres per minute corrected for atmospheric pressure, and the scale marked accordingly. The quantity passing through the circuit was varied for different tests by the use of spring clamps on the rubber connections between the Crowell pump and the meter. In this way a series of marks were obtained on the scale in terms of litres per minute. The accompanying diagrammatic sketch, Fig. 8, shows the arrangement of the apparatus used :—

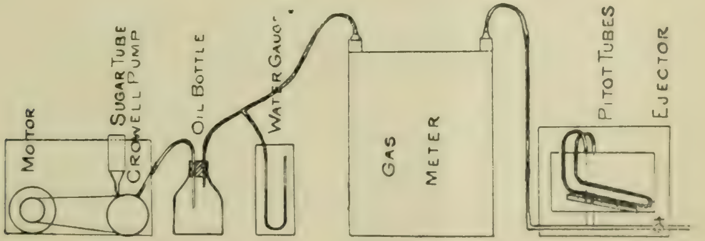


Fig. 8.

Sketch showing Arrangement of Apparatus for Calibrating Manometer.

As a further check a small galvanized sheet-iron tank 2 ft. diam. and 2 ft. deep, having 1-in. inlet and outlet cocks, also a $\frac{3}{8}$-in. copper tube let in at the tap for connection to the ejector, was set up. The tank was also fitted with a gauge glass and a centimetre scale on the outside in order to show the level of the water. The tank was filled with water and the outlet cocks turned on, the inlet air passing through the pitot tubes of the ejector to the tank. The quantity of water displaced was measured by means of the centimetre scale and the time noted by stop watch. The displacement of water in the glass gauge tubes of the ejector was noted and marked on the scale. Owing to the fact that the tank was small, the quick loss of head meant a variable velocity pressure, so that the position of the menisci in the glass gauge tubes varied considerably. The process was therefore reversed and the rate of air displacement measured by filling the tank with water. By this means a satisfactory check on the marks up to 35 litres per minute was obtained. The water pressure available was insufficient to give a higher rate of displacement than this. However, as the results of these tests checked the marks obtained by the gas meter tests, and as the displacements of the menisci checked up against the curve given in the Report of the Technical Commission of Inquiry, sufficient information was available to fill in the blank spaces on the pitot tube scale.

In order to obtain satisfactory results from the closed tank method, a tank of large sectional area should be used, and it should be set at a height of 12 or 15 ft. above the work bench so as to minimise the effect of variable velocity pressure due to loss of head.

Discussion on Part I.

C. H. J. Clayton congratulated Messrs. Warren and Read on the very comprehensive survey of methods and apparatus.

Konimeter.—Regarding the Konimeter—Broken Hill members were more or less *au fait* with the original instrument as used in South Africa, and the modification of microscope and konimeter as one instrument was, in his opinion, rather a questionable advantage for an instrument which must be taken underground. It was quite difficult enough to count dangerous particles of dust up on the surface without adding further difficulties. The cheaper and more rugged the instrument going down the shaft the better, as the majority of places where sampling was urgently needed were very difficult of access, such as winzes, rises, etc., and the more compact and small the outfit the less knocking about it would get. Slides were easily carried, and a number of spots, quite proportional to the number of places which could be visited before coming up to the surface again, could be readily taken.

Gravimetric Sampling.—Dealing with the sugar tube, Crowell pump collector :—They have all viewed with inward misgivings the spectacle of the Technical Commission conducting tests which cost a considerable sum of money, apparently quite incommensurate with the amount of information afforded, and it always seemed that the money could be much better expended and the energy much better utilized in taking a rough cheap indicative sample, and spending the balance on dust-allaying gear and improved machinery when it was necessary.

Read Water-spray and Kotze Ejector.—The description of the Read Water-spray and Ejector was particularly valuable, affording as they apparently did, a means of carrying out gravimetric estimations with a reasonable expenditure.

General.—The summary of all the methods will be extremely valuable for the use of the mining student, for, of late years, he must be a little fogged as to what a mining engineer's duties really were—the production of metal in such an economical manner that he would be enabled to sell it or putting all his energies into channels that had very little to do with metal production.

They were all very much indebted to Messrs. Warren and Read and to the South Mine for affording them the opportunity of carrying out their investigations, and for the very valuable information, carefully sifted and weighed as it has been. They have saved many weary hours of reading and better still many hours of fruitless endeavour expended in wrong directions. A more comprehensive paper has not been placed before the Branch for some time, and the authors deserve heartiest congratulations.

Mr. Warren, in reply, dealing with the use of the microscope added to the konimeter in underground work, said that he had found the microscope a useful addition, as it enabled a determination to be made on the job as to whether an atmosphere was very dusty or otherwise ; one could not expect accurate counts of dust-spots to be made in that way, but he had not found the necessary equipment at all cumbersome.

Part II.

Records of Tests and Efficiencies of Methods.

This section of the paper contains a record of tests which have been conducted for the following purposes :—

(1) Comparison of Read water-spray sampler with sugar tubes; also two tests showing comparison with electrostatic method as a matter of interest.
(2) Tests to show the efficiency of the Read water-spray sampler.
(3) Efficiency tests of the konimeter.

The water-spray sampler was that described in Part I., and illustrated in Fig. 3.

Brass sugar tubes made to the U. S. A. Bureau of Mines dimensions as illustrated in Fig. 6, Part I., were used.

The sugar was the best grade white sugar obtainable, graded to pass 20 mesh and remain on 80.

The electrostatic apparatus, which was assembled on the mine, consisted of the following parts :—

(1) An oil insulated transformer to raise the pressure from 110 or 220 to 22000 volts.
(2) A Coolidge X-ray tube to act as a valve ensuring that only uni-directional current be supplied.
(3) A collecting tube of aluminium $2\frac{3}{8}$ in. diam. and 12 in. long.
(4) A glass rod $\frac{1}{4}$ in. diam., suspended centrally within the aluminium tube. Two copper wires 18 swg. are wound spirally round the glass rod serving as an electrode.
A small fan driven by 1-20th h.p. motor to aspirate the dust-laden air through the collecting tube. The collecting tube is made positive to the electrode and is grounded for safety.

The konimetric spots were obtained with an instrument of the type illustrated in Fig. 2, Part I.

All samples were obtained on surface in the crushing section of the mill.

Two ejector units were used, the samples being taken simultaneously.

The following is a comparison of the results obtained simultaneously by—

A. Read water-spray sampler.
B. Sugar tubes.

A. Read water-spray apparatus	45 litres per minute	All samples 31 mins.
B. Sugar tube (brass)	32.5 ,, ,, ,,	

No.	Vol. of Air sampled in cub. m.	Weight of Dust in milligrams.			Mg. per cub. m. of Air.	Remarks.
		Gross.	Sugar Blank.	Net.		
83 A	1.36	14.00		14.00	10.20	The arithmetic average of these results in terms of milligrams dust per cubic metre of air
B	1.00				10.60	
84 A	1.395	18.00		18.00	13.00	
B	1.00				14.00	
85 A	1.395	3.00		3.00	2.20	
B	1.00				Nil	
87 A	1.395	18.16		18.16	13.02	
B	1.000				13.10	
88 A	1.395	7.24		7.24	5.20	
B	1.000				5.33	
89 A	1.395	10.70		10.70	7.70	=Water-spray sampler
B	1.000				6.30	
90 A	1.395	4.10		4.10	2.94	
B	1.000				3.50	=6.68
91 A	1.395	12.40		12.40	8.90	
B	1.000				8.20	
92 A	1.395	5.80		5.80	4.16	Sugar tube=6.78
B	1.000				4.00	
93 A	1.395	6.97		6.97	5.00	
B	1 000				6.60	
94 A	1.395	7.50		7.50	5.40	NOTE.—These results are not representative of mill working places, but were purposely secured over a wide range of values.
B	1.000				5.00	
100 A	1.395	17.0		17.00	12.20	
B	1.000	15.3	1.7	13.60	13.60	
101 A	1.395	3.2		3.20	2.30	
B	1.000	4.9	1.7	3.20	3.20	
102 A	1.395	2.0		2.00	1.40	
B	1.000	3.2	1.7	1.50	1.50	

DISCUSSION OF RESULTS.

In conducting these tests great care was taken to ascertain as accurately as possible the weight of dust in the sugar by means of running blanks.

It will at once be seen that where the total weight of dust may only be 4 or 5 mg. this is a matter of great importance. It must be admitted that the method of using sugar suffers from the defect that the amount of dust present as impurity is variable and cannot be estimated with absolute certainty. However the arithmetic average of the results of 14 pairs of samples, giving 6.68 for the water-spray and 6.78 for the sugar tube, seems to indicate that both methods are equal in efficiency, but that the water-spray method is preferable for routine work on the grounds that the filtering medium requires no assay office treatment.

The following tests were run to compare the water-spray sampler—sugar-tube method and electrostatic method :—

No.	Vol. of air sampled in cub. m.	Gross weight of dust in mg.	Sugar blank.	Net weight of dust in mg.	Mg. dust per cub. m. of air.	Remarks.
98 A	1.35	3.3	—	3.3	2.44	A=Water-spray
B	1.35	5.4	1.7	3.7	2.65	B=Sugar tube
C	6.72	13.3	—	13.3	1.98	C=Electrostatic
99 A	1.395	3.7	—	3.7	2.65	
B	1.000	5.5	1.7	3.8	3.80	
C	6.83	20.1	—	20.1	2.93	

These tests were merely run as a matter of interest. It will be noticed that the results do not vary greatly.

EFFICIENCY TEST OF WATER-SPRAY SAMPLER.

Weighed quantities of silica dust, which had been ignited to get rid of the water of combination, were fed to the apparatus as shown in diagrammatic sketch, Fig. 9.

The Nessler tube, its fittings and the dust were thoroughly dried in a water bath for two hours. The Nessler tube containing the weighed quantity of silica dust was attached to the inlet air nozzle of the water-spray sampler.

Four tests are here recorded in which varying quantities of dust were used. The weight of dust left in the Nessler tube is not included in estimating the recovery of the sampler—

No.	Weight of Dust in Milligrams.				Percentage Recovery.
	Fed to Nessler Tube.	Balance in Nessler Tube.	Entering Sampler.	Recovered from Sampler.	
1	75	2.7	72.3	72.3	100.00
2	100	5.6	94.4	88.3	93.53
3	25	8.4	16.6	15.5	92.17
4	55	11.5	43.5	41.7	95.86
				Average	95.39

Statement of reported efficiencies of various methods, calculated on a Weight Basis :—

Method.	Efficiency per cent.	Reference.
Sugar tube ..	70-98	Fieldner, Katz and Longfellow, Report Tech. Paper No. 278, U. S. Bureau of Mines, 1921.
Palmer water-spray	45	Fieldner, Katz and Longfellow, Journal of Industrial Hygiene, Sept. 1920.
	82, as compared with electrostatic 50, 69	Smythe and Izard, Journal of Industrial Hygiene, Sept. 1921. Bill—Journal of Industrial Hygiene, November 1919.
Read water-spray	95	
Electrostatic ..	82.3	Bill—Journal of Industrial Hygiene, November 1919.

TESTS OF THE EFFICIENCY OF THE KONIMETER.

Gravimetric samples were secured with the Read water-spray appliance and sugar tube simultaneously. Konimeter spots were taken at regular intervals while sampling and subsequently counted, an average count being thus obtained for the duration of the sample.

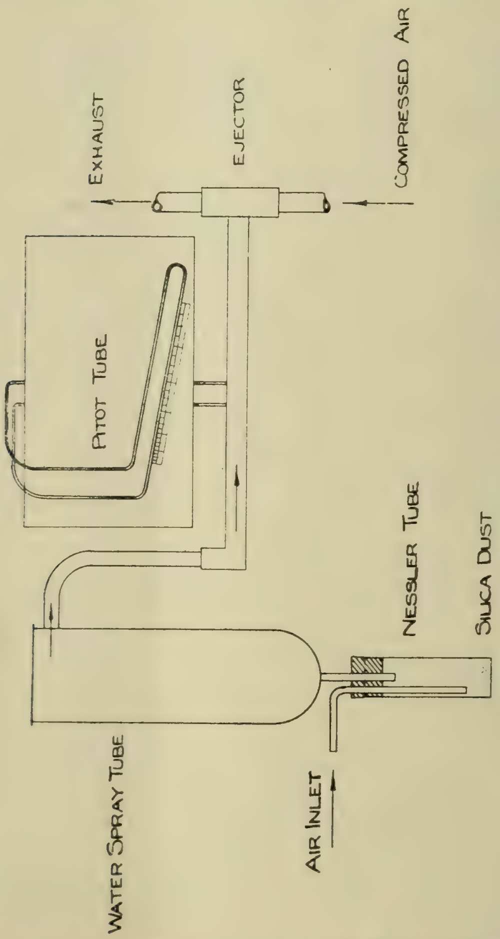


FIG. 9.

ARRANGEMENT OF APPARATUS FOR EFFICIENCY TEST OF WATER-SPRAY SAMPLER.

The gravimetric samples obtained per medium of the water-spray apparatus were made up to 500 cc. by the addition of distilled water.

50 cc. were set apart and six counts were run on this portion. A portion of the solution was placed in a Thoma haemocytometer cell having a cubic capacity of .004 cc. The area of the cell was measured in terms of eyepiece diaphragm squares, and the number of particles in the cell counted in the manner already described in Part I. in connection with konimeter spots. The six counts obtained in this way were averaged for each sample in terms of particles per cc. of air sampled.

The haemocytometer count obtained for the water-spray sample was taken at 100% and the konimeter counts compared.

Haemocytometer counts were not run on the sugar-tube sample, as these figures would be useless owing to the insoluble dust present as impurity in the sugar. The object in securing the sugar-tube sample was simply to check the gravimetric result given by the water-spray apparatus.

The following schedule gives the results obtained:—

No.	Vol of Air sampled in cc's.	Dust Particles per cc. of Air sampled.		Comparative Efficiency.
		Haemocytometer av. of 6 counts.	Konimeter av. of 6 counts.	
1	1,500,000	2,220	220	10
2	1,500,000	975	92	9.4
3	1,550,000	3,444	815	23.6
4	1,395,000	2,200	280	12.7
5	1,395,000	4,441	709	15.9
6	1,395,000	5,732	1,709	29.8
7	1,000,000	2,288	343	15.8
8	1,285,000	4,497	1,035	23.0
			Average	17.4

From this investigation it would appear that the efficiency of the konimeter is low.

The quantity of solution examined in the haemocytometer cell is so small in comparison to the total volume that a large number of counts should be run on any sample, and in any case there is no guarantee that the dust particles are evenly distributed throughout the solution; indeed, individual counts on the same sample varied considerably.

The writers regard the konimeter as a serviceable instrument providing a quick and ready means of determining the relative rather than the absolute dustiness of the air. It is also of the greatest value in providing a quick means of examination of dust particles in respect of physical character such as shape and size.

SUMMARY.

The following is a brief summary of the points of interest in this paper :—

(1) Routine gravimetric dust sampling may be successfully carried out in places difficult of access such as mine workings.

(2) Gravimetric samples may be successfully done by means of filtering the air through sugar in tubes, provided (*a*) the technique of the operation be standardised on the lines laid down by the U. S. A. Bureau of Mines ; (*b*) the fact that sugar contains variable quantities of insoluble dust as impurity increases the liability to error and adds to the amount of work necessary in the assay office.

(3) The Read water-spray sampler provides a useful and efficient means of collecting dust samples ; whereas the field work is no more difficult than in the case of sugar tubes, the liability to error in the laboratory treatment of the sample is considerably minimised.

(4) Comparative tests with sugar tubes and Read's water-spray sampler show both methods to be equal in efficiency.

(5) The Kotze ejector pump provides a very convenient means of aspirating the air, and a sugar tube or water-spray appliance is interchangeable on the unit described in Part I.

(6) The konimeter is not a highly efficient scientific instrument but is very useful as a quick means of determining the relative dustiness of air.

The writers' acknowledgements are due to Mr. W. E. Wainwright and Mr. A. Fairweather, General Manager and Mine Superintendent respectively, of the Broken Hill South Limited, for the facilities afforded in making these investigations ; to Messrs. Thomas, Black and Hooper for assisting in the design of the Kotze ejector pump unit and for providing the illustrations which accompany this paper ; to Mr. E. H. Gilbert for assembling and operating the electrostatic unit.

DISCUSSION.

Mr. Thos. G. Hanton said he would like to thank Messrs. Warren and Read for the very interesting paper they had presented on the "Determination of Dust Suspended in the Air," in which they had given a comprehensive account of the subject from the time when it first became of local significance, and had also embodied the results of the more recent developments of their own experimental work on the Broken Hill South Ltd. Judging from the figures and the information supplied, this work must have been on a very extensive scale. To those members who had not been in touch with the question of dust determination the first portion of the paper would be especially valuable in affording a résumé of the earlier work in that direction, both locally and elsewhere. The results given, showing the comparatively low quantitative efficiency of the konimetric method appeared to substantiate the objections raised, during the discussion of Mr. Warren's paper "Mechanical Methods of Allaying Dust," of 31st May, 1921, to the accuracy of that method for the absolute measurement of suspended dust. The reliability of the means described for determining the efficiency of the konimeter, seemed to depend entirely upon the uniformity of the mixture of the gravimetric sample with the 500 cc. of distilled water into which it was put. Any inequalities in the dust contents throughout the 500 cc. would be reflected in the 50 cc. withdrawn and in the final haemocytometer samples. The authors stated that variations occurred in haemocytometer counts from the same sample. That would seem to indicate the possibility of unequal distribution of dust throughout the water into which it was placed, assuming that in all the samples the process of counting would not introduce another variable. The final haemocytometer sample appeared to bear to the original 500 cc. somewhat the same relation that the 5 cc. of the konimeter sample did to the air of the place sampled. On the Zinc Corporation, the experience had been that the conditions giving rise to a series of low konimeter counts had invariably shown low gravimetric readings, and similarly high konimeter counts had been followed by high gravimetric results. The konimeter, despite its acknowledged inherent defects, still retained its field of usefulness as pointed out by the authors, as an instrument for the rapid determination of the relative order of the dust contents of the air, and for the examination of the size and physical characteristics of the particles. From its convenience and small size it was well adapted to underground work. The cumbrous nature and awkwardness of handling of the various types of gravimetric apparatus, as well as the necessity for some power means of aspirating the air (except a hand pump be used for this purpose), had been amongst the factors militating against the adoption of that method for underground routine dust-sampling. Locally the most convenient form of power available for the purpose was compressed air. Even in those mines which had a lighting circuit underground, the necessity for advance wiring and the trouble involved in conducting electrical power to the more remote and less easily accessible working places, would render the use of that form of power for routine sampling both costly and cumbersome. Mr. Read was to be congratulated on the design of a convenient and apparently most efficient form of apparatus for gravimetric sampling, in which many of the objectionable features of the older types had been eliminated. The equipment described—the Read water spray and the Kotze ejector—appeared to represent a very marked advance in every direction over any earlier gravimetric sampling outfit. By its use the taking of gravimetric dust samples should be greatly facilitated. The thanks of all those interested in the subject were due to the authors for the information which they had made available, both in the results of their tests and in the description of the apparatus developed.

Mr. J. W. Salter said Messrs. Warren and Read had demonstrated very clearly the many advantages of using the Read water spray in preference to the sugar tube in obtaining gravimetric samples. There being no sugar to dissolve and filter, and no impurity in the dust-arresting medium, necessitating a blank being run, the operations in the laboratory should be more rapid. With regard to the authors' tests showing the relative efficiency of the konimeter and Read water-spray apparatus, it seemed hard to believe that the efficiency of the former was only 17% of the later, and he thought that further investigation should be done along these lines. In making gravimetric tests, a low result, say 3 mg. per cub. m., did not necessarily imply that the air was satisfactory, as there may be a large number of particles of very fine dust in that air, which did not possess much weight, but were the more harmful portions of the dust. Also in the collaring of holes where gravimetric results generally showed high, that was due mainly to large particles, which were less harmful. The dust spots of the konimeter would show these particles under the microscope. When series of tests were taken with the konimeter in conjunction with either the sugar tube or water-spray methods of sampling, a factor could be worked out to give an average number of particles per cc., which would be the equivalent of 1 mg. per cub. m. and when that was obtained, the konimeter could be used for testing the mine air without the necessity of always doing a parallel gravimetric test. That was very handy in testing out-of-the-way places like winzes or rises, where it was awkward to get the cumbersome gravimetric apparatus. During the course of the investigations by the Technical Health Commission and whilst sampling was in progress in the underground workings of the North Mine, various factors showing the relation of konimeter counts to actual weight were obtained from a large number of tests.

These factors were as follows :—

In stopes after firing ..	138	particles per cc.	=1 mg. per cub. m.
In winzes after firing ..	128	do	do
In winzes during boring operations	122	do	do

Those factors had been applied in subsequent tests and it was claimed gave from 90 to 95% of the actual weight to be obtained by running a gravimetric test. By using factors like these, a large number of tests could quickly be done with the konimeter which would give the operator all the data he required without resorting to the slower gravimetric method, also there was no necessity to get the absolute weight of the material, unless an analysis was required. and the information obtained from examining dust spots under a microscope was far more valuable than that obtained from the gravimetric method.

The konimeter factor could be checked at intervals against the actual weights obtained by, say, the Read water-spray, and then for ordinary routine work used solely by itself.

W. D. **Whiteman** said in Series I. all tests were done using the Kotze ejector as the aspirating medium. Additional tests using sugar tubes on both the Crowell pump and ejector units or sugar tubes on Crowell unit as against Read water-spray sampler on the ejector unit, concurrently would, he thought, be interesting in a comparative sense, seeing that all the Commission tests had been carried out by the Crowell pump method of aspirating. Although both air meters were of the pitot tube type the rating method would be checked. An additional advantage that appeals, from the results of tests by the Read water-spray sampler, was that nearly 1-3rd more mine air could be tested in a given time, which would be a considerable saving in time over the sugar-tube method where larger samples were necessary for chemical analysis of the dust contents.

Series III., Efficiency Tests on Konimeter.—With regard to these tests he would have expected a much higher efficiency than the 17.4% average as compared with the haemocytometer tests from the Read water-spray samples, and the results of the gravimetric tests Nos. 1-8, taken along with that series of tests would be interesting to show the relationship existing between gravimetric and konimetric results. Certain relationships had been recorded from South African experience, e.g., as quoted in a Bulletin of the Transvaal Chamber of Mines :—

1 mg. of dust per cub. metre=160 particles per cc. of 2.5 microns av. diam.
1 ,, ,, ,, ,, ,, ,, =130 particles per cc. of 2.75 microns av. diam.
1 ,, ,, ,, ,, ,, ,, =100 particles per cc. of 3.00 microns av. diam.

A definite relationship was also noted in the tests carried on at Broken Hill. If in any mine a ratio, such as the above, could be established for various processes, such as firing in stopes, pop firing, firing in development ends (winzes, rises, drives, etc.), and also with regard to boring operations in the same places, the scope of the konimeter in spite of any decreased efficiency, he thought could be widened, without risk of error as to the amount of dust that was safe in a working place. Its use in that way would lessen the number of gravimetric samples required to be taken, and economise in time. Returning to the efficiency tests where the samples were taken on the breaker floor and in the belt conveyor tunnels, it was mentioned that there were "big particles of dust which were damp," and it was quite possible that those particles consisted of aggregations of very fine particles of dust. These may have been drawn into the konimeter and deposited on the vaselined slides as such, thereby tending to lessen the count, while in the Read water-spray sampler the fine particles might become dissociated and show a higher count when examined in the haemocytometer cell. The chances of error would practically be the same as regards the count whether in haemocytometer cell or on konimeter slide, seeing that the former would deal with dust from about 12 cc. of air sampled, as against 5 cc. or 10 cc. as the case may be in the latter, so that any deficiency in the konimeter method would have to be accounted to the imperfect deposition of dust on the vaselined slide, provided the specified amount of air was being drawn through the konimeter. The authors had made due allowance for the possibility of error creeping in, due to the extremely small size of the sample taken for the count as compared with the total sample taken from the water-spray sample, viz., 1 : 125000. In conclusion he wished to compliment Messrs. Warren and Read on the valuable research work they had done in connection with the furtherance of knowledge concerning gravimetric sampling as it affects dust suppression, as evidenced by the papers just read before the Institute.

MR. C. H. J. CLAYTON said he had to thank Messrs. Warren and Read for the valuable statement of efficiencies, and the way in which their work had been summarised. Regarding the electrostatic apparatus he personally preferred to leave such a method to the physicist until it was perfected sufficiently to remove it from the realm of scientific to that of commercial interest. With simpler apparatus at their disposal and commercially of quite a sufficient efficiency, he did not think that the efficiency of the electrostatic method concerned the mining engineer. Nothing could be more satisfactory than the comparison of sugar tube and Read water-spray apparatus, and the figures justified the claims put forward for the Read water-spray as an efficient dust-sampling machine. The konimeter efficiency tests were hardly of the same order as they were so completely at variance with the practical work already carried out in mining operations. The efficiency varying from 9.4 to 29.8 was not reflected in comparisons taken underground while the Technical Commission were conducting their investigations. During those investigations it appeared that if a working place were satisfactory with konimeter counts it would also be quite satisfactory with sugar tube results. The chief value of the konimeter appeared to be the differentiation of satisfactory and unsatisfactory conditions; when a spot was unsatisfactory the konimeter did not give the exact degree, but was that necessary from the mining man's point of view? If they were to insist upon more perfect conditions would they eliminate any ailments? He doubted it, for probably a fresh crop would spring up elsewhere. Conditions underground and on the surface varied so much from point to point of time, that it hardly seemed worth while to strive for greater efficiency in dust counting. In some places the konimeter gave questionable results and the Read water-spray would then be available to decide the question. From the information contained in the paper it would appear that the ordinary konimeter counting, with the Read water-spray and Kotze ejector as a check when required, would give all that was required in mining and milling operations.

Messrs. P. H. Warren and T. A. Read, in replying to the discussion, said they desired to express their appreciation of the generous spirit in which the paper was received. There were one or two points raised by members upon which a little further elaboration might be expended. They had one object in presenting the paper which they would again like to stress—that was the need for standardisation in all the operations incidental to dust sampling. Where time and money were spent in collecting samples it was well worth while doing the work in such a way that reliable and comparable results were obtained. The point raised by Messrs. Hanton and Whiteman concerning the uniformity of the water mixture from which haemocytometer counts were made, was always present in that sort of work. The authors noticed that when the suspensions were allowed to stand for any time the dust settled out, and it was impossible to again obtain an even mixture. To guard against error from that source they made their counts as soon as possible after collecting the samples. They also noticed that the dust was not evenly distributed over the bottom of the cell; to counteract that they estimated the number of dust particles in terms of diaphragm squares in the eyepiece and counted a complete section right through the cell. Unfortunately, in any such method of counting the ratio of the volume of the cell to the total volume of the suspension in water was very large, and a small error in the average count of the cell would make a very large difference in the total number of particles;

they therefore exercised every possible care to try and arrive at correct averages in the work of counting. Mr. Whiteman also referred to the possibility of aggregations of fine dust particles appearing as one big particle on a konimeter slide, while similar aggregations would separate out in a water solution. They agreed that the efficiency of the konimeter did suffer on that account. A greater number of small dust particles from .5 to 1 microns in size were clearly visible in the haemocytometer cell than was the case on the konimeter slide. They did not make the claim that the comparative efficiency of the konimeter was 17.4% as shown by their tests, but merely that the efficiency appeared to be low. They did not understand Mr. Clayton's statement that "the konimeter efficiency tests were so completely at variance with the practical work already carried out underground." The fact that a large number of konimeter counts had been obtained which yielded similar results for similar operations merely meant that the efficiency of the konimeter, whatever it might be, did not vary much in different instruments. They had never heard that any practical mining operation was capable of proving that the konimeter could effect a hundred per cent. catch. Further, the fact that during the investigation of the Technical Commission, low konimeter counts accompanied low gravimetric results obtained by sugar tube, merely showed that, for some of the operations, there was a more or less constant relationship —a relationship which, while fairly constant for operations resulting in a preponderance of particles below 5 microns in size, broke down when the number of large particles increased unduly. It might not be generally known that one of the officials of the U. S. Bureau of Mines had, as a result of recent investigation, tentatively placed the efficiency of the konimeter at about 13%. They agreed with Mr. Salter that a factor could be worked out showing a relationship between particles per cc. as obtained by the konimeter and milligrams per cubic metre as obtained by gravimetric sampling. The factor, however, would vary with each set of conditions met with ; for instance, the factor for boring with piston drills would differ widely from that obtained from dry-head jackhamers ; such features as the physical character of the dust and ventilating conditions would also have an influence. The statement, for example, that 128 particles per cc. corresponded to 1 mg. per cub. m. in winzes after firing, was of no value unless accompanied by information as to how long after firing, whether a water blast was used or not, and what were the ventilating conditions. Mr. Whiteman suggested that they give the relationship of konimeter counts to the results of the gravimetric samples of those tests quoted in connection with the comparative efficiency of the konimeter. The following were the factors of tests 1 to 7 :—

Number.	Factor in Particles per cc. to 1 mg. per cub. m.
1	27
2	15
3	9
4	45
5	15
6	52
7	24

It would be seen that those factors did not resemble those put forward by Mr. Salter. That was probably due to the fact that the samples were secured in the crushing section of the mill and contained a large amount of particles which were considerably over 5 microns in size, and also the high efficiency of the water-spray sampler would tend to reduce the ratio. The authors also were of the opinion that to correctly interpret the result of a gravimetric sample it should be accompanied by a konimeter count, so that the investigator might note the physical character of the dust. They were glad to have had an opportunity of placing the results of the investigation before members of the Institute, and they were greatly indebted to those members who prepared discussion.

PROCEEDINGS AUS. I.M.M. (INC.)] [N. S., NO. 47, 1922.

VOLUMETRIC DETERMINATION OF IRON: NEW METHOD OF REDUCTION.

BY P. F. THOMPSON.

INTRODUCTORY.

THE reduction of iron from the ferric to the ferrous state prior to its titration by permanganate or dichromate may give erroneous results in the presence of other reducible metals such as titanium. Stannous chloride and metallic zinc both reduce tetravalent titanium to a lower form, and so increase the consumption of the oxidizing standard solution. Sulphuretted hydrogen and sulphurous acid do not reduce titanium, but are slow in action and troublesome to manipulate.

Sulphuretted hydrogen is considered to be the best reducing agent for iron in the presence of titanium.* In the method to be described the advantages of the H_2S reduction are retained without the disadvantages of using the gas. The reduction is carried out by the action of a metallic sulphide having a solubility product of such a value that the sulphide ions are utilized as fast as they are formed, and but little free sulphuretted hydrogen is given off.

Cadmium sulphide was the first to be tried, but its solubility product (3.6×10^{-29})† was apparently too low. The reduction was satisfactory, but the excess was very difficult to remove. Zinc sulphide (S.P. $= 1.2 \times 10^{-23}$) was found to be quite satisfactory in both respects. The zinc sulphide is preferably used in the form of an emulsion prepared from pure zinc sulphate as follows:—About 100 grm. of $ZnSO_4 7H_2O$ is dissolved in 2 litres of water. H_2S is passed until no further precipitate is obtained, the acid formed being barely neutralized from time to time by the addition of ammonia. The precipitation of the sulphide in a slightly acid solution gives a finer emulsion than that prepared in an alkaline solution, and any iron present remains dissolved. The precipitate is allowed to settle and the clear liquid decanted. The precipitate is then shaken up with more water, acidified with a little sulphuric acid, allowed to settle, and the liquid decanted as before. Water is added to bring the volume to about 2000 cc. This emulsion keeps indefinitely, and only requires shaking to be ready for use.

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* Treadwell-Hall, "Quantitative Analysis," 1919, p. 609.

† Treadwell-Hall, "Qualitative Analysis," 1916, p. 21.

PROCEDURE FOR PERMANGANATE TITRATION.

The sulphuric acid solution containing the iron to be titrated is placed in a short-necked flask and heated to boiling. Sufficient sulphide emulsion to give a decided milkiness is added slowly, and the boiling continued until no reaction is obtained with a drop of thiocyanate on a spot-plate. If the concentration of sulphuric acid is not too great, complete reduction is obtained very quickly, not the slightest colour being given with sulphocyanide.

If the sulphur liberated by the reaction is brownish in colour, small amounts of heavy metal sulphides are present, and should be filtered off before removing the excess of zinc sulphide. If, however, the residue is white or faintly yellow, some dilute sulphuric acid and a piece of pure, white marble is added, and the solution boiled vigorously until the sulphur becomes granular and the solution quite clear. By this time the slight excess of sulphuretted hydrogen should have been removed, but it is preferable to make quite sure by testing the vapour with lead acetate paper. The flask is then cooled quickly under the tap, and the cold solution titrated in the usual manner with standard permanganate solution.

A blank titration, using the same amounts of acid, zinc sulphide, and marble, should be made. Not more than 0.1 cc. of decinormal permanganate should be required.

PROCEDURE FOR DICHROMATE TITRATION.

If the solution containing the iron is strongly acidified with hydrochloric acid, the greater part of the acid should be neutralized before reducing with the sulphide. The emulsion is added to the boiling solution as before, but the progress of the reduction is indicated by the disappearance of the yellow colour of the ferric chloride. On account of the greater solubility of the marble in the hydrochloric acid, care must be taken that the ebullition does not become too vigorous. A stream of carbon dioxide from a Kipp apparatus may be used instead of the marble, or it may be dispensed with since the titration can be carried out hot there is little chance of oxidation by the air taking place. The solution is titrated with decinormal potassium dichromate, using a somewhat weaker ferricyanide solution than usual as indicator. The comparatively small amount of zinc in solution does not appear to interfere with action of the indicator.

A blank experiment should, of course, be conducted, as in the case of the permanganate titration.

EXPERIMENTAL.

TITRATION OF A FERRIC SOLUTION AND THE EFFECT OF TITANIUM.

A solution of ferric sulphate acidified with sulphuric acid was prepared. The amount of iron in this was determined by measuring out 50 cc. with a standard pipette and precipitating the iron as hydroxide with ammonia after adding a little paper pulp.* The precipitate was filtered, washed, ignited strongly, and weighed as Fe_2O_3. The weight was 0.1482 grm., which corresponds to 0.1037 grm. of iron.

The following are the results of titrations of this solution, taking 50 cc. and reducing with 20 cc. of the zinc sulphide emulsion :—

Ferric Solution.	Titanium Sulphate Soln. added.	$\frac{N}{10}$ $KMnO_4$ (c.c.)	= Grm. of Fe.
(*a*) 50 cc. ..	nil.	18.55	.1038
(*b*) 50 cc. ..	nil.	18.50	.1037
(*c*) 50 cc. ..	nil.	18.50	.1037
(*d*) 50 cc. ..	½ grm. TiO_2	18.52	.1037
(*e*) 50 cc. ..	½ grm. TiO_2	18.53	.1037
(*f*) 50 cc. ..	1 grm. TiO_2	18.55	.1038
(*g*) 50 cc. ..	1 grm. TiO_2	18.55	.1038
(*h*) nil. ..	1 grm. TiO_2	.05	—

A trace of iron was found in the titanium solution. This evidently accounts for the slight increase when the titanium was added.

COMPARISON OF METHODS OF REDUCTION.

A solution of pure ferric alum was prepared. This was titrated with $KMnO_4$ after reduction with (*a*) electrolytic zinc and (*b*) zinc sulphide. It was also titrated with dichromate after reducing with stannous chloride and zinc sulphide respectively.

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* Washington, "Chemical Analysis of Rocks," 1919, p. 154.

50 cc. of the solution was taken in each case, and the iron was determined in the same amount by precipitation; 0.2099 grm. of Fe_2O_3 were obtained, equivalent to .1469 grm. Fe.

Titration.	Reduction.	cc. Standard.	Iron Found.
			Grm.
Dichromate	$SnCl_2$	26.1	0.1461
,,	ZnS	26.0	0.1456
Permanganate	Zinc	26.2	0.1467
,,	ZnS	26.3	0.1472

For these titrations the standard solutions were both checked by standard thiosulphate, the latter being compared with decinormal hydrochloric acid by means of the iodide–iodate reaction. The hydrochloric acid was standardized by calc-spar.*

A sample of hæmatite iron ore was handed to eighteen students for analysis, using both zinc and zinc sulphide reduction. The averages of their reports were :—Zinc reduction, 55.3 per cent. iron; and zinc sulphide, 55.4 per cent.

A standard sample of titaniferous iron ore was analyzed by this method. It contained 60.49 % of iron and considerable titanium; 0.25 grm. was taken in duplicate, requiring (*a*) 27.10 cc. and (*b*) 27.08 cc. of decinormal permanganate. These are equal to (*a*) 60.48 and (*b*) 60.43 % of iron respectively, after allowing 0.1 cc. for the blank.

The foregoing results show that the reduction of iron with zinc sulphide is at least as accurate as the older reduction methods. The reagent can readily be prepared in a state of purity, and its efficiency does not alter on keeping. The method provides a rapid and reliable procedure, which is of particular value to the chemist who prefers the permanganate, but who uses the hitherto more expeditious dichromate method for the determination of iron even when some titanium is present.

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* Masson, "Jour. Soc. of Chem. Industry of Victoria," vol. i., pp. 62–71.

Papers and Discussions.

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SCREENING TESTS ON FINE SCREENS.

BY C. C. FREEMAN AND H. J. HOOKINGS.

(Read before the Broken Hill Branch, 12th September, 1922.)

UNDER a contract relating to the sale of a Broken Hill concentrate, the differentiation between granular and slime concentrate is determined by the proportion of the material which will pass a 200-mesh (I.M.M. standard) screen, and this has led to considerable attention being devoted to the question of laboratory sizing tests with screens of this order.

It was found that many of the screens turned out by the makers during the war, and also since its termination, did not conform with the I.M.M. specifications as closely as some supplied in pre-war days. A perfect screen is hardly to be expected, bnt when the above contract was framed the parties anticipated that, by special care being taken during manufacture, a screen at least as good as the best pre-war screens would be available. This, however, has not proved to be the case, and the large irregularities in screens recently supplied render them useless for sizing determinations under the contract. The makers were approached through the Institution of Mining and Metallurgy, London, with the view of obtaining a few screens conforming as closely as possible to the 200-mesh I.M.M. standard, and in response to a request twelve pieces of screen were supplied. As the results of an examination of these screens may be of interest to other members of the profession, the authors present the following résumé of their investigations.

It was apparent from the outset that the screens were very far from perfect, and the measurements of the apertures give an idea of the number of oversized holes in each screen.

Screens Nos. 1 to 6 were examined along two lines at right angles to another, and the number of all holes over .0635 mm. along these lines were noted. An ordinary 200 I.M.M. screen, recently supplied by makers, was similarly examined. The results are shown in Table I.

Screens Nos. 7 to 12 were examined by another operator in the same way, but in this case only openings measuring approximately .079 mm. and over were counted. For comparison purposes an ordinary 200 I.M.M. screen (not the same as the one referred to in Table I.) was examined. These results are set out in Table II.

Tables I. and II. show that the largest variations occur in one direction, referred to as " with the warp," but in the other direction a very large number of holes are over .0635 mm. but under .079 mm.

The gauge of the wire used in these screens appears to be quite correct—viz., .0635 mm. diam.—but the irregularity of the weaving is shown by counts of the numbers of apertures and wires in successive lengths of 1.59 mm. along the lines examined.

According to the I.M.M. standard, in a length of 1.59 mm. the number of apertures plus wires should total 25, whereas the counts show the figures set out in Table III. This is rather interesting in that the results obtained by screening tests (referred to below) vary according to the figures shown in counts with the woof in Table III.

In order to try and obtain a screen more closely approaching the correct I.M.M. specification, screen No. 10 was cut up, and on a portion of this the holes of largest dimensions were blinded with shellac. This screen is referred to as " Screen A." Actual screening tests show that this allowed particles having two dimensions up to .079 mm. to pass the screen, so that some apertures even in this screen were well above the true 200 I.M.M. standard. Screens 1 to 9 and 11 to 12 were mounted, and a sample of zinc concentrate prepared so as to contain sufficient fine material to give about 40 % passing the screens under examination. This sample was carefully mixed

TABLE I.

MICROSCOPICAL EXAMINATION OF SCREENS NOS. 1-6.

Measurements taken across opposite parallel sides of each screen.

A—with warp. B—with woof.	Screen No. 1.		Screen No. 2.		Screen No. 3.		Screen No. 4.		Screen No. 5.		Screen No. 6.		Ordinary 200 I.M.M. supplied 1921, No. "B."	
	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.
Total number of holes ..	1533	1532	1582	1592	1516	1562	1586	1557	1560	1543	1542	1544	1573	1552
No. of holes over .0635 mm. and under .079 mm. ..	193	297	120	178	138	119	151	105	143	67	132	80	281	253
No. of holes .079 mm. and under .095 mm.	76	47	46	38	50	28	57	15	56	6	48	11	218	65
No. of holes .095 mm. and under .111 mm.	18	nil.	12	nil.	24	1	19	nil.	18	nil.	16	nil.	64	1
No. of holes .111 mm. and under .127 mm.	2	nil.	2	nil.	1	nil.	1	nil.	2	nil.	1	nil.	12	nil.
No. of holes .127 mm. and over	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	7	nil.
Reducing the above to percentage of total holes we have :—														
% of oversized holes.. ..	18.9	22.5	11.4	13.6	14.0	9.5	14.4	7.7	14.0	4.7	12.8	5.9	37.0	25.0
% over .0635 mm. and under .079 mm.	12.6	19.4	7.6	11.2	9.1	7.6	9.5	6.7	9.1	4.3	8.6	5.2	17.9	16.3
% .079 mm and under 095 mm.	4.9	3.1	2.9	2.4	3.3	1.9	3.6	1.0	3.6	0.4	3.1	0.7	13.6	4.2
% .095 mm. and over ..	1.3	nil.	0.9	nil.	1.6	nil.	1.3	nil.	1.3	nil.	1.1	nil.	5.2	nil.

N.B.—Correct aperture for 200 I.M.M. screen = .0635 mm.

TABLE II.

MICROSCOPICAL EXAMINATION OF SCREENS NOS. 7–12.

Measurements taken across screens about 2 in. from edge of same.

A—with warp. B—with woof.	Screen No. 7.		Screen No. 8.		Screen No. 9.		Screen No. 10.		Screen No. 11.		Screen No. 12.		Ordinary 200 I.M.M. supplied 1921, No. "C."	
	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.	A.	B.
Total number of holes (theoretically 1575) . .	1560	1561	1573	1640	1529	1512	1570	1597	1571	1574	1571	1615	1551	1492
No. of holes app. .079 mm. and under .095 mm. . .	124	23	100	10	83	78	96	16	55	72	119	11	202	29
No. of holes .095 mm. and under .111 mm.	13	nil.	17	nil.	11	nil.	12	nil.	8	nil.	12	nil.	45	nil.
No. of holes .111 mm. and under .127 mm.	5	nil.	2	nil.	3	nil.	1	nil.	1	nil.	nil.	nil.	15	nil.
No. of holes .127 mm. and over	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	1	nil.	6	nil.
Reducing the above to percentage of total holes we have :—														
No. of holes app. .079 mm. and under .095 mm. . .	8.0	1.5	6.4	0.6	5.4	5.2	6.1	1.0	3.5	4.6	7.6	0.7	13.0	1.9
No. of holes .095 mm. and under .111 mm.	0.8	nil.	1.1	nil.	0.7	nil.	0.8	nil.	0.5	nil.	0.8	nil.	2.9	nil.
% of holes .111 mm. and under 127 mm.	0.3	nil.	0.1	nil.	0.2	nil.	0.1	nil.	0.1	nil.	nil.	nil	1.0	nil.
% of holes .127 mm. and over	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	nil.	0.1	nil.	0.4	nil.

N.B.—Correct aperture for 200 I.M.M. screen = .0635 mm.

and divided into a number of equal parts of 200 grm. each. A test of each screen was made by placing the dry sample on the screen (without a rubber, etc.) and hand-shaking until nothing further passed the screen. This operation required about 1½ hours' shaking. The undersize from each of these tests was collected and the operation repeated on the small screen "A." The results were determined by weight of material remaining on the screens, and any loss in the operation is assumed to be in the undersize. The results of these tests are shown in Table IV.

It will be noted that the figures in the final column—*i.e.*, corrected in terms of screen "A," vary from 36.2 % to 37.9 % through the screen, and these may be considered to be within the experimental limit of error for this type of work, particularly in view of the fact that the screens and mountings were new.

Table IV. shows that, with these "special" screens, screening results of duplicate samples vary from 37 % through with No. 8 screen to 43.5 % through with No. 9 screen. Reference to Table II. shows that, whilst the number of oversize holes with the warp are substantiallly the same in these two cases, the number of holes about .079 mm. long with the woof is much greater in the case of No. 9. This difference is also very apparent in Table III. Results are also given of screening tests using a screen (marked "B," and probably of pre-war manufacture), which was one of a number supplied prior to 1919, and also with a screen (marked "C") from a number supplied since that date. The former is a much better screen than the ordinary post-war screens manufactured by the same makers, and compares favourably with the average of the 12 "special" screens, although not so good as No. 8. It will be noted that No. 9 is as bad as that marked "C."

Conclusions.

As it is understood that special care was used in weaving these screens, it would appear as if screens cannot be made to closely conform with specifications. The correct designation of a screen is determined by the actual size of the largest particle passing through same, and this is dependent on the size of the largest aperture in such screen. After close examination of these screens the writers would suggest that it might be possible to obtain screens woven to, say, 220 mesh, wherein the accidental larger apertures would more nearly conform to the 200-mesh specifications.

Apparently one larger piece of screening was cut up to make the 8-in. squares, but the large variations disclosed by Tables I. to IV. indicate the need for close examination of all screens before they are used. The makers' specification cannot be accepted as final.

By blinding the largest holes (which can be picked out with the naked eye) the screens can be improved, but such a large portion of each screen is affected by oversize holes that it is almost impossible to fully correct a screen to true specifications.

The above clearly shows the need for standardization in regard to fine screens used for sizing purposes.

The authors' thanks are due to Mr. S. H. Sloan, who conducted all the screening tests recorded in Table IV., and to Mr. W. R. Miller, for assistance in counting and measuring of apertures.

TABLE III.

Number of lengths 1.59 mm. wherein the number of apertures plus wire totalled :—

Screen.		21.	22.	23.	24.	24½.	25.	25½.	26.	27.	28.
No. 7 screen	A.	1	4	10	17	14	48	6	22	2	—
	B.	—	—	2	37	25	43	13	7	—	—
No. 8 screen	A.	1	1	10	18	7	48	13	27	4	—
	B.	—	—	—	—	—	8	15	75	22	7
No. 9 screen	A.	1	2	9	17	17	36	16	26	1	—
	B.	—	3	45	22	4	8	7	26	7	2
No. 10 screen	A.	—	3	8	15	20	35	21	22	4	—
	B.	—	—	—	7	15	37	36	18	10	3
No. 11 screen	A.	—	3	8	18	14	41	18	21	4	—
	B.	—	1	7	26	10	33	16	19	11	—
No. 12 screen	A.	—	2	10	10	15	41	21	24	2	—
	B.	—	—	—	2	11	31	25	36	21	—
Ordinary 200 I.M.M. screen, supplied 1921—No. "C"	A.	1	5	8	36	—	56	—	22	1	—
	B.	—	—	50	63	5	7	—	—	—	—

A, with warp. B, with woof.

N.B.—Number of apertures plus wires per 1.59 mm. should be 25.

TABLE IV.

Screening tests on sample of zinc concentrate.

Column No. 1 shows percentage on the screen after hand screening (dry) until nothing further passed—approximately 1½ hours. The undersize was then re-screened on Screen A and column No. 2 shows percentage retained on this screen. Column 3 is the sum of columns 1 and 2 and shows the result of the test in reference to screen A. Column 4 shows percentage "through" in terms of screen A.

——	1.	2.	3.	4.
Screen No. 1	58.9	4.2	63.1	36.9
,, ,, 2	60.0	3.8	63.8	36.2
,, ,, 3	61.3	2.3	63.6	36.4
,, ,, 4	61.9	1.2	63.1	36.9
,, ,, 5	61.4	2.4	63.8	36.2
,, ,, 6	59.9	2.9	62.8	37.2
,, ,, 7	59.9	2.5	62.4	37.6
,, ,, 8	63.0	0.6	63.6	36.4
,, ,, 9	56.5	5.6	62.1	37.9
,, ,, 11	59.9	2.6	62.5	37.5
,, ,, 12	61.1	2.4	63.5	36.5
"B" (pre-war)	60.3	2.0	62.3	37.7
"C" (supplied 1921)	57.0	6.1	63.1	36.9

DISCUSSION.

T. A. READ said he must congratulate the authors on their work in checking up the 200-mesh I.M.M. screens, and also for pointing out the defects of the so-called standard 200-mesh I.M.M. screens. To those of us who have had to check up screens, the amount of work described by the authors and their assistants is recognized. The investigations made by the authors were all the more valuable at the present time, when all zinc concentrate sales from Broken Hill were governed by a limiting percentage of the material passing through a 200-mesh I.M.M. standard screen, and he was of the opinion that the term "mesh" in the contracts should be abandoned, and the term "maximum aperture of .063 mm. substituted. By doing this the manufacturers of screens could insert extra wires to take care of the oversized aperture.

PROCEEDINGS AUS. I.M.M. (INC.)] [N. S., No. 48, 1922.

THE GOLD-ANTIMONY VEINS OF COSTERFIELD, VICTORIA

BY F. L. STILLWELL, D.Sc.

I.—GENERAL DESCRIPTION.

THE township of Costerfield is situated in the parish of Costerfield, county of Dalhousie, and lies 6 miles east of Heathcote and 30 miles south-east of Bendigo. It is separated from Heathcote by the Mt. Ida Range, which forms the western boundary of the Silurian formation. It is situated at the head-waters of Tin-pot Creek, a small southerly tributary of the Deegay Ponds Creek, which flows east to the Goulburn River.

The field was discovered in 1860 by two prospectors, Coster and Field, who were guided there by a shepherd, M'Nicol. Coster and Field had already discovered an antimony lode about three miles nearer Heathcote, when M'Nicol offered to guide them to the larger outcrop. The main Costerfield reef at that time projected above the surface, and was white through oxidation, though the bluey sulphide showed on fracture. No alluvial gold of any consequence was obtained at Costerfield, but a little was obtained at Redcastle, 7 miles to the north, and in some of the gullies to the east.

Following its discovery there came a period of prosperity to Costerfield, while the mining operations were confined to the oxidized zone. The early success of mining at Costerfield was due to the gold values, and it is probable that the oxidized zone was secondarily enriched. The earliest available record of values is a crushing of Coster and Co. on the Antimony reef (now known as the Costerfield reef), when 200 tons from a depth of 70 ft. averaged 4 oz. per ton in July, 1861. At this time a crushing mill had not yet been erected at Costerfield, and the ore was carted and treated at Redcastle. The massive antimony ore, consisting of mixed oxides and sulphides, was separated by hand-picking as "marketable ore," and the remainder crushed for gold. A higher average is recorded from South Costerfield in September, 1861, when a 15 cwt. parcel of ore, obtained by Morris and Co. from a depth of 45 ft. on the Kilburn reef, averaged 9 oz. 16 dwt. per ton. The Costerfield reef was, however, the main producer, and up till March, 1866, averaged about 1 oz. per ton on a weekly tonnage of 50. In the subsequent years the gold recovery was much lower. The following table illustrates the early production of antimony ore in Victoria,* and more than 80 % of this was obtained from the Costerfield mines. It also gives the tonnages and gold yields from Costerfield,† but occasional returns are not recorded, and the figures are, therefore, not complete :—

Year.	Production of antimony ore in Victoria.	Costerfield gold production.	
	Tons.	Tons.	Oz.
1862 1863	807	—	—
1864	578	2464	2280
1865	729	3031	2592
1866	435	2950	1945
1867	272	1107	577
1868	841	4131	1026‡
1869	709	658	333
1870	1662	1658	531
1871	1575 (1278 from Costerfield)	1793	652
1872	1573 (1518 ,, ,,)	1644	546
1873	1428 (1194 ,, ,,)	1605	475
1874	588	634	246
1875	1750	567	893
1876	2529	1432	524
1877	2640	1974	996
1878	2628	860	886
1879	496	168	108
1880	334	307	151
1881	70	—	—
1882	376	—	—
1883	35	—	—
Total	.. 22,055		

These figures are sufficiently accurate to illustrate the higher gold values obtained in the earlier years. They are also shown in a graph

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* Min. Stat. Vic., 1864–1883, Vic. Parl. Papers.

† Rpts. of Min. Surveyors and Registrars, Vic. Parl. Papers.

‡ Including 743 from tailings.

(Fig. 1) of the Victorian production of antimony ore and Costerfield gold, in which the antimony ore in tons is represented by a full line and the gold in ounces is represented by the dotted line. From 1903 to 1920 Costerfield has produced the whole of the Victorian production, and the graph shows the tons of antimony concentrates recorded by the Mines Department in these years, which are comparable to the tons of "antimony ore" in the earlier period. The gold curve in the later period is plotted from these tonnages combined with the average gold values furnished by the smelter, deductions being made for the difference between gross and net tonnages.

The gold production of the early days decreased as mining operations approached the sulphide zone, where the miners were faced with the metallurgical problem of the recovery of the gold from the antimony. Not more than 20 % of the gold in the reefs is amenable to amalgamation with mercury, and various attempts to recover the remainder by smelting were unsuccessful. Nevertheless, the antimony production increased, benefiting in the early and late seventies by the high market for antimony caused by European wars. During the latter period (1875–78) small tonnages of rich gold ore were obtained from the Alison shaft. Prices then slumped, and production practically ceased in the beginning of the eighties.

As a result of the early operations a large quantity of rich tailings accumulated. These were estimated by a mining manager in 1866 to be worth ½ to 3 oz. of gold per ton, and they were re-treated in 1868 for the first time. The losses were still considerable, and large heaps remained containing 6 % Sb and 7 dwt. and over of gold per ton. Various efforts were subsequently made to recover these, without success.

A more prosperous era commenced in 1903 with the advent of the Bombay Syndicate Ltd., which changed into the Victorian Syndicate Ltd. and then developed into the present Costerfield Mining Company. This company's success has depended principally on the solution of the smelting problem at the smelting works of a sister company at St. Helens, near Liverpool, where the gold is completely recovered from the antimony. A contributing factor has been the gradual elimination of losses in separating the ore from the mullock in the crushing and concentrating mill, and the present mill residues at

Costerfield contain, on an average, only 15 gr. of gold per ton and $\frac{1}{2}$ % of antimony. The period of this company's operations is brought into strong contrast with the earlier period by Fig. 1, in which the relative positions of the gold and antimony curves are completely reversed.

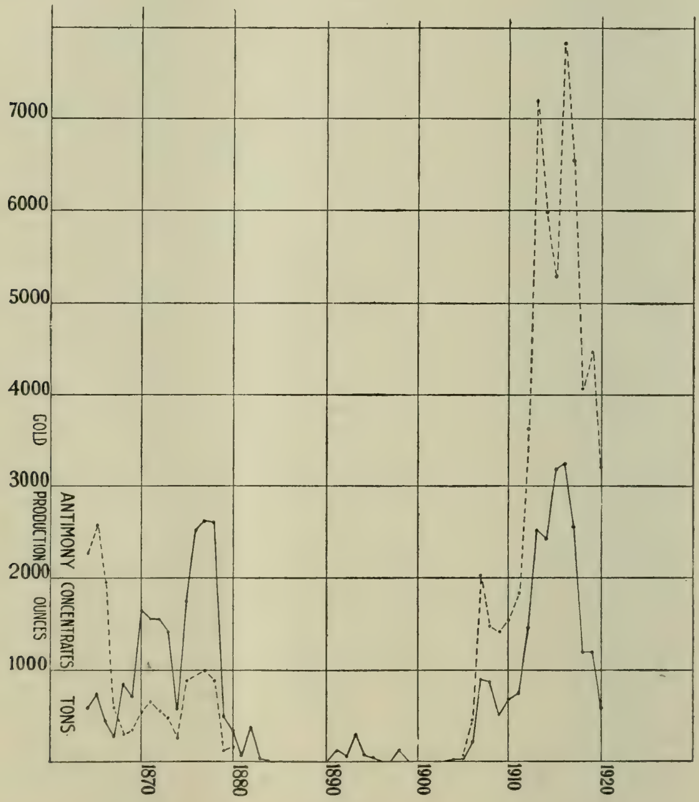

Fig. 1.

Graph of the Production of Antimony Concentrates in Victoria (full line) and the Costerfield Gold Production (dotted line).

The mine products as shipped at present to the smelter are :—

1. Hand-picked ore, averaging 60 % Sb, 2 oz. 12 dwt. of gold per ton.

2. Coarse concentrates, averaging 63 % Sb, 2 oz. 15 dwt. of gold per ton.
3. Fine concentrates, averaging 45 % Sb, 4 oz. of gold per ton.
4. Flotation concentrates, averaging 50 % Sb and 1 oz. 12 dwt. of gold per ton.

These average values extend over a period of fifteen years, and apply to the ore mined at various depths between 420 ft. and 780 ft. at the main mine. The ore obtained from the Alison mine, at South Costerfield, one mile south of the main shaft at Costerfield, has been of lower grade, the hand-picked ore averaging 43 % Sb and 1.4 oz. of gold per ton, owing to a relatively high percentage of iron pyrites.

Up to the end of 1905 the total production of antimony ore in Victoria is recorded in the statistics of the Victorian Mines Department as 22,972 tons, valued at £177,736. This tonnage has been practically equalled by the present company. The quantities of finished ore that have been received at St. Helens from Costerfield up to December, 1920, are shown in the following statement compiled at St. Helens :—

Year.	Tons.	Sb. %	Gold. Oz. per ton.
1906	281, average	45	2.00
1907	1095, ,,	47	2.25
1908	865, ,,	47	1.75
1909	575, ,,	47	3.00
1910	496, ,,	47	2.50
1911	840, ,,	48	2.60
1912	1320, ,,	50	2.60
1913	2390, ,,	52	3.00
1914	2630, ,,	46.4	2.58
1915	2900, ,,	45.0	1.75
1916	3100, ,,	52.3	2.53
1917	460	—	—
1918	51, average	57.4	2.69
1919	3350, ,,	57.8	3.03
1920	1593, ,,	57.5	3.38

Total .. 21,946 dry weight ; average, 50.7 % Sb., 2.603 oz. of gold.

In May, 1917, the export of the ore was suspended through lack of shipping accommodation. Production continued for a while at a diminished rate, and stocks of ore accumulated. After the close of the European War export was resumed in the early part of 1919, and the stocks reduced. Production has been reduced during the year (1921) on account of the low market combined with high freights and high costs.

The production and cost of production for a period of five years are illustrated in the following table :—

Year.	Mined ore.	Concentrates.	Production cost.	Upkeep.	Administration.	Development.	Total cost per ton.		
	Tons.	Tons.	£	£	£	£	£	s.	d.
1914	7,797	2432	22,038	3488	1201	2533	12	1	0
1915	11,454	3061	31,840	4161	1499	8972	15	3	0
1916	10,235	2882	39,401	3430	3163	5558	17	18	0
1917	13,673	2636	35,688	4066	4138	3859	18	2	0
1918	4,962	1143	16,451	3177	1335	78	19	5	0

The fundamental reason for the gradual increased cost per ton throughout this period has been the rising price-level of wages and materials, suffered by all industries.

The number and value of the antimony reefs that have already been discovered at Costerfield have established the field as an important antimony belt. It extends to Redcastle, seven miles north of Costerfield, where auriferous antimony has been mined. At South Costerfield, one mile south from the main line along the lines of strike, the Alison shaft has been an important point of production. Two miles further south of the Alison shaft is Bradley's claim, from which antimony ore has been obtained. There are four equipped shafts on the field, including the Alison shaft, but for some time operations have been confined to the main shaft. The shafts are :—

Main or north shaft	1015 ft. deep.
Minerva shaft	295 ft. ,,
Bombay shaft	340 ft. ,,
Alison shaft	400 ft. ,,

II.—GEOLOGY.

1. *The Country Rock.*—The mines are situated in an area of Silurian mudstones, which strike about 25° W. of N. and pitch, at Costerfield, at a varying angle up to 15° N. No Ordovician rocks have been met with in depth, as inferred by Gregory.* Gregory based his inference on Ranft's report † that abundant Ordovician graptolites occur in the waste heap of the Alison mine at South Costerfield. This report is not correct, and is probably founded on the numerous darkened areas which resemble worm-tracks in the mudstone. The only fossils so far observed in this waste heap are fragments of crinoid stems.

About 2 miles west of Costerfield the rocks are alternating beds of highly silicified quartzites and shales, dipping vertically or at a high angle to the west, and still further west are the basal or marginal conglomerates of Mt. Ida. Sandstones associated with these rocks contain abundant casts of Silurian fossils. The Heathcotian series of igneous rocks occurs at Heathcote, west of Mt. Ida, and 6 miles west of Costerfield, while southward, at Tooborac, are large outcrops of granodiorite.

The Costerfield mudstone is massive and dense, fine-grained, and of even texture, with no marked cleavage. It consists essentially of fine particles of quartz, minute flakes of mica, and kaolin. Rutile needles have been observed, and also occasional grains of tourmaline and zircon. The secondary minerals present include pyrite, arsenopyrite, and widespread areas of granular carbonate. There is at times a tendency towards the development of cleavage in the neighbourhood of some of the veins and faults, which may perhaps account for the use of the term "schist" in the report of the Mines Inspector in 1867.‡ Slight differences in composition sometimes produce a banding parallel to the bedding, and this is the only guide to the bedding throughout the whole of the mine workings.

The beds have been gently folded into a broad, low anticline, which trends to the westward in depth. The anticlinal fold is exposed in the main crosscut at the 580-ft. level, west of the shaft, but has not been reached in the main crosscuts at the 420-ft., 685-ft., and 786-ft. levels. At these levels it can only be observed in the westerly branch crosscuts off the south drives. In some of the mine crosscuts the beds are almost horizontal, and they rarely dip at a higher angle than 45°, though the dip may be increased in the neighbourhood of faults. In the main crosscut at the 985-ft. level, however, the beds at the shaft possess a dip of 80°, decreasing to 60° in the present face of the crosscut 60 ft. from the shaft. The centre of the anticline will be 200 ft. or more west of the shaft at this depth.

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* "The Heathcotian: a Pre-Ordovician Series and Its Distribution in Victoria." J. W. Gregory, Proc. Roy. Soc. Vic., vol. xv. (n.s.), part 11, p. 168.

† "Origin and Formation of Auriferous Rocks and Gold." J. A. H. T Ranft, Sydney, 1889.

‡ "Goldfields of Victoria." R. Brough Smyth, p. 417.

In the immediate neighbourhood of the reefs the mudstone is at times indurated and impregnated with iron pyrites and arsenopyrite. The crystals of pyrites are small, and never so abundant as in most of the mineralized slates and sandstones of the Bendigo goldfield. Pyrite and arsenopyrite crystals are sometimes intergrown, and are usually surrounded by thin border zones of quartz and white mica. The mica in some rims resembles the mica of the ground-mass of the mudstone, but at other times it is transformed into chlorite. The hardening of the mudstone is irregular, and chiefly due to the size and number of the isolated and secondary areas of carbonate. Sometimes there is a development of secondary quartz as well as the carbonate.

The mudstone has been penetrated to a depth of 1000 ft., and displays a remarkable uniformity in composition. There is no general hardening or any other general change in the rocks in this depth, as first suggested by the Mines Inspector in 1867.* The hardest rock in the experience of the miners is reported from an east branch crosscut off the north drive on the Costerfield reef at the 420-ft. level. A specimen of the mudstone from this crosscut only differs from the typical mudstone in possessing slightly larger grains of quartz, but the amount of argillaceous matter present is too large to permit its description as a fine-grained sandstone.

2. *Faults.*—When the bedding is relatively flat the massive mudstone is traversed by three sets of joint planes dividing the rock up into cuboidal masses. The lateral joints usually pitch north at the same angles as the bedding, and as soon as a thin film of pug is found on a joint-plane the latter is identified as a plane of movement and becomes a fault. The course of the faults is only indicated by

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* *Op. cit.*, p. 417.

the seam of pug or crushed rock, and, as there are no guiding characteristics in the bedding, its displacement is difficult to measure.

The main series of faults appear to be a set of west-dipping faults which, in many instances, pitch north with the strata. One of these, known as Whitelaw's fault, has been conspicuous in mining operations, and has a pitch of about 15° N., and a westerly dip varying from 25° to 55°. It intersects the main ore-body (Kendall's reef) at 660 ft. S. at the 585-ft. level, and displaces it about 6 ft. It has been followed northwards through the stopes for about 300 ft., when it dips below the 686-ft. level. At the 686-ft. level the displacement of the reef has increased to 28 ft. Between the 686-ft. and 786-ft. levels the reef is carried by this fault and other similar breaks to a distance of 200 ft. west at the lower level.

In addition to this class of west-dipping strike-faults are the oblique faults, of which the more important trend approximately N.W. One was intersected in a branch crosscut 900 ft. S. at the 420-ft. level, dipping 47° to the N.E., and carrying 12 in. of quartz of no value. This fault intersected the Costerfield reef, and an enrichment in both antimony and gold occurred at the intersection. A subordinate set of oblique faults trend approximately N.E.

3. *Relation of Reefs and Faulting.*—Quartz-veins of no value, and up to 3 ft. wide, are often observed to lie along fault-planes. Such veins are clearly younger than the faulting.

The quartz and stibnite veins are frequently observed to meet a fault-plane, then turn and follow it for a short distance before pursuing their normal courses. A diagrammatic example is shown in a branch west crosscut at the 686-ft. level (Pl. I., fig. 1), where a vein of quartz and stibnite, 3 in. wide, is intersected by two small faults. The vein has the appearance of being truncated by the faults, but the ends are connected by veins of irregular and varying width. In every case examined the stibnite veins turn and follow the fault planes in this manner. Traces of stibnite and quartz are found along the fault, and at the 786-ft. level some of the veins have maintained their

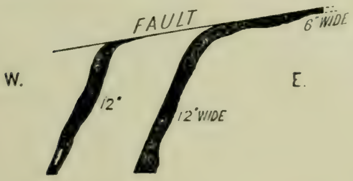

Fig. 2.
Sketch of Two Reefs of Stibnite exposed in Mason's Drive, 500-ft. South at the 786-ft. Level Main Shaft.

average width along the fault-plane and have consequently been stoped out (Fig. 2).

It is therefore clear that the veins, in general, are younger than the faults. The mineral solutions, in their upward course, have met the pre-existing fault-planes and have been diverted for short distances by them.

III.—Reefs.

The mineral veins at Costerfield can be separated into two types —(1) the quartz–stibnite and stibnite–quartz veins, which carry gold, and (2) quartz-veins which are uniformly unproductive of antimony or gold. The productive reefs are included in the first type, while the laminated quartz-veins and the fault reefs belong to the second class.

1. *Stibnite Reefs.*—The stibnite-bearing veins are in general more or less vertical, and cut across the low-angle bedding in the region of the anticline. They are simple lodes underlying both to the east and west, and any individual vein appears to be limited in the direction of its strike. In the aggregate they form a branching system with an average strike about N. 20° W., and the strike of individual veins may show considerable variation in both directions. The payable reefs average in width about 6 in., but may open out into bulges 24 in. wide or more and thin out in places to 2 in. or less. In many cases there is a definite wall to the vein, but the bulgy, lenticular character is generally more pronounced in the stibnite reefs than in the quartz reefs. The form of a stibnite vein below the 686-ft. level and below Whitelaw's fault is shown in Fig. 3, where the bulges occur immediately below joint planes. In another instance certain bedding planes affect the reef in like manner.

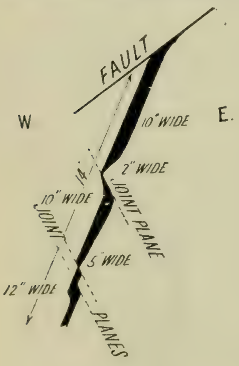

Fig. 3.
Form of Reef in Jury's Stopes, below Whitelaw's Fault at the 686-ft. Level.

A transverse section (Fig. 4) through the mine in a plane 430 ft. south of the main shaft illustrates the system of productive veins. As most of the mine workings are south of the shaft, a cross section through the main crosscuts from the shaft does not reveal the position

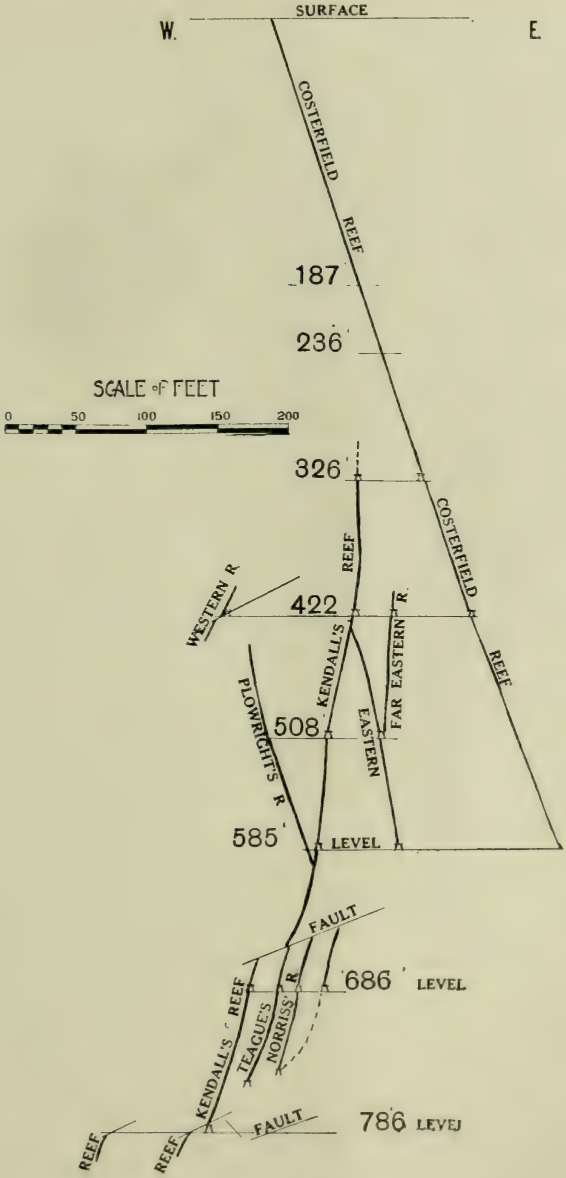

FIG. 4.

COSTERFIELD TRANSVERSE SECTION, 430-FT. SOUTH OF SHAFT.

and character of all the reefs. At the 420-ft. level the vein belt has been proved for a width of 220 ft. On the western side a stibnite vein underlies 57° west, and intersects beds, dipping 35° west, while the main Costerfield reef occurs on the eastern side, underlying 70° east, where the beds are dipping 35° east. The section illustrates the displacement of the main or Kendall's reef by Whitelaw's fault between the 686-ft. and 585-ft. levels. From the 686-ft. level four veins have been stoped upwards towards the fault. They do not continue as payable veins above the fault, and appear to be the downward continuation of Kendall's reef below the fault. Kendall's reef has been stoped through from the 786-ft. level to above the 420-ft. level, and throughout this height maintains a well-defined wall. Opposite the main shaft at the 420-ft. level Kendall's reef is represented by a few insignificant threads, but southwards, opposite the Minerva shaft, it appears to have been stoped to the surface in the early days.

In the stopes on Kendall's reef, between the 585-ft. and the 686-ft. levels, a branching vein, now known as Plowright's reef, was followed into the hanging wall. Plowright's reef has not been worked in the plane of the section, but the stopes at the 508-ft. level are within 50 ft. of it. It is a reef with an eastern underlie, and has now been stoped upwards to within 20 ft. of the 420-ft. level. A western branch crosscut from the south drive on Kendall's reef at the 420-ft. leve was extended to cut the upward extension of Plowright's reef, and proved it to be a very thin thread at this height, but the continuation of this crosscut intersected another west-dipping fault and the new payable " Western " reef with a western underlie.

Between the 420-ft. and 585-ft. levels a branching vein entered the footwall of Kendall's reef, and has been worked as the " Eastern " reef down to the 585-ft. level over a length of 670 ft. In the plane of the section it became poor and unpayable below the 585-ft. level. At the end of the 585-ft. south drive, 1050 ft. S., the Eastern reef has proved to be a valuable ore-body between the 585-ft. and the 686-ft. levels. An offshoot from the Eastern reef is the " Far Eastern " reef, with a very slight underlie to the west. No enrichment or increase in size occurs at the junction of these veins.

In plan the reefs form a branching system just as in transverse section. Norris's reef, traced southward, runs into the east branch of Kendall's reef. Teague's reef, traced southwards, joins the west branch of Kendall's. The east and west branches of Kendall's then continue southwards, and finally unite. Further south again Kendall's reef meets a branch of the Eastern reef. Meanwhile, other veins have developed to the east and west, and are disclosed in branch crosscuts.

The Costerfield reef, which was the main producer in the early days, was rich opposite the main shaft, where it was worked down to the 420-ft. level and followed for another 200 ft. by an underlie shaft. In shallow depths it has been followed north for 550 ft. and south for 1400 ft. All the profitable work, however, on Kendall's reef and associated veins has been confined to that part of the mine commencing about 300 ft. south of the shaft. There is no change in the character of the rocks in this part of the mine, and it is probably due to the fact that between the 585-ft. and 786-ft. levels this length of ground is fractured by Whitelaw's fault and other faults beneath it. As these faults pitch north they will come into the cross section opposite the shaft at lower levels, and on this account it is possible that this payable zone will then be nearer the shaft. A recent development in this direction has occurred in the main west crosscut at the 885-ft. level, where a vein with an eastern underlie of 72° has been met carrying 6 in. of stibnite and payable values. At the same time the western portion of the anticline has not been tested opposite the main shaft at the higher levels where payable veins may exist.

2. *Mineral Composition of Stibnite Reefs.*—The reefs are composed chiefly of quartz, stibnite, and gold. A carbonate, probably ankerite, is present, and under the microscope an occasional crystal of zinc-blende is seen. The freedom of the stibnite from arsenic and other impurities is remarkable, and the assay values of the hand-picked ore show only .04 % of arsenic. The pyrite and arsenopyrite are confined to small crystals in the wall-rock and to the neighbourhood of occasional slaty seams in the veins. They are collected in the fine concentrates in the mill, which contain, on an average, about 3 % of arsenic. Crystals of stibnite have occasionally been seen in the wall-rock, but it is evident that the wall-rock has been permeable to the pyrite and arsenopyrite and substantially impermeable to the stibnite. Pyrite is a more noticeable impurity in the Alison reefs at South Costerfield.

In the most profitable sections the reefs consist of massive stibnite assaying up to 3 oz. of gold per ton and over. Such a reef may be 12 in. wide, and usually shows a thin selvage of quartz on both sides, which is often no more than $\frac{1}{4}$ in. wide. Small crystals of quartz, frequently idiomorphic, are actually distributed through the apparently solid stibnite, and a little quartz is distributed in irregular patches through the vein. In places the quartz forms a suture seam down the middle of the vein. This seam of quartz is usually attached to a thin, dark, fragmentary film of slate which is more or less replaced, and passes into residual laminæ associated with mica. It may be partly lined with a film of carbonate and impregnated with pyrite and arsenopyrite. Particles of slate have also been observed embedded in massive stibnite and partly impregnated with veinlets of stibnite.

If free gold is observed in the reef it is more usually found around the quartzy areas.

The stibnite usually forms a dense, steely mass, but at times assumes a coarsely crystalline character. The needle antimony or stibnite in prismatic needles generally becomes more common as the percentage of quartz increases. In the transition areas from massive stibnite into quartz the stibnite assumes the form of a reticulating system of veinlets through the quartz. Some of these veinlets consist of a layer of minute needles embedded in quartz.

There is every gradation between stibnite reefs with a little quartz and quartz reefs with a little stibnite. As the stibnite veins are followed in length and height they eventually become poor either because they dwindle in size or else because they become too quartzy. The Costerfield reef at the surface consisted mainly of stibnite and its oxidation products, and was reported in 1867 to be from one to ten feet wide. Where now seen in the end of the 420 ft. north drive it is a well-defined quartz vein. 4 in. wide, containing gold and antimony. Where seen in the branch crosscut at the 585-ft. level it shows 6 in. of antimony, with 1 to 2 in. of laminated quartz on either side. Norris's reef has become small and unpayable at the intermediate level between the 686-ft. and 786-ft. levels, but at the same time a number of small leaders have developed in the footwall (Pl. I., fig. 2). Some of these small veins consist wholly of quartz, and some consist of quartz with small lenses of antimony. Others consist of stibnite with a small amonut of quartz, and others, again, consist of massive stibnite with the typical quartz selvage.

The gold in the reefs consists of (1) free yellow gold, of (2) free rusty gold, and (3) combined gold.

The free yellow gold occurs embedded in the quartz, and will amalgamate with mercury. Some reefs, which are poor in antimony, are payable on account of the free gold. The free gold of rusty or copper-red colour is quite distinguishable from the yellow gold in the quartz, and generally occurs in the stibnite. Its colour is only skin deep, being normal on the fractured surface. Gold of this colour, though free, will not amalgamate with mercury, and it is believed to be coated with a film of sulphide. It is often near small vesicles, and its appearance is spongy or cellular, which suggests that a certain amount of stibnite has been removed, leaving secondary gold behind. An occurrence has been noted in the south drive, 1050 ft. S., at the 686-ft. level, where veins of needle antimony reticulate through quartz showing free yellow gold, and in these veins the cellular rusty gold forms the base or centres of the radiating needles. In many other specimens the rusty gold appears where the antimony assumes the crystalline prismatic structure. It therefore seems possible that this structure may have developed from the dense, steely stibnite and been accompanied by the separation of gold. No average increase in gold values occurs in those portions of the stibnite veins where the free rusty gold is observed. It is known that if the massive gold-containing stibnite is melted that the prismatic needle antimony may result, which is practically free from gold. The same result appears to have been achieved in nature in the production of rusty gold.

Experience has tended to lead the management to the belief that where the needle antimony is abundant in the reef the gold values are likely to be low. A recent occurrence tends to illustrate this point on a large scale and in an unprecedented manner at Costerfield. At a distance of 1050 ft. S. the Eastern reef has been followed from the 585-ft. level by a winze and traced to the 686-ft. level. Its average width in the winze was 6 in., and assays proved the gold content up to 5 oz. per ton and rich to within 25 ft. of the 686-ft. south drive. The latter drive in this part of the mine had followed an unusually large reef, consisting of 3ft. of quartz and 4 ft. of massive stibnite, which assayed 4 dwt. and 2 dwt. of gold per ton. It forms the first known record of massive stibnite with low gold values at Costerfield. The decrease in gold content in the 686-ft. south drive appeared to commence beyond a small oblique fault. It can be noticed that the ore with the low values appears to be more coarsely crystalline than the general run of ore. Crystal faces ½ in. wide and 2 in. long are showing in a small hand specimen, and the stibnite prisms lie more or less in parallel position. Small idiomorphic crystals of quartz can be detected lying between the prisms of stibnite, and free gold has been observed in this ore with a pocket lens. The low grade ore persisted in the drive for over 1000 ft., when the character of the ore gradually changed, and rich gold values appeared. Free yellow gold appeared in the quartz, and free rusty gold appeared in the dense massive stibnite, which varied in thickness from 6 in. to 15 in.

Apart from all the free gold visible in the reefs, there is a considerable amount of gold contained in the massive stibnite, whose presence is revealed by assay. Ranft * has suggested the presence of a silicate of gold, but no mineral with physical properties differing from known minerals has been discerned in the massive stibnite, so that it is unlikely that there is a compound of gold present. The gold is probably either dissolved in the stibnite or else it exists as free, dark, interstitial particles too minute for ordinary microscopical observation.

A comparison of the two sets of average values of the mine products in the accompanying table has some bearing on the manner of occurrence of gold in the reefs. The first set covers a period of four months in 1917, and the second set covers a period of eight months in 1920–21. During the latter period the average grade of the ore was below normal, as the tonnage was very restricted, and included the low-grade ore from the south drive at the 686-ft. level. During 1920 the concentrating mill was enlarged, partly reconstructed, and a tube mill installed. Prior to the installation of a mineral separation machine the mill had been designed to avoid the production of slimes. Now the final crushing is much finer and the values are recovered from all the slimes.

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* "Origin and Formation of Auriferous Rocks and Gold." J. A. H. T. Ranft, Sydney, 1889, p. 19.

Year.	Hand-picked ore.			Coarse concentrates.			Fine concentrates.			Slimes.		
	Sb.	Au.	As.	Sb.	Au.	As.	Sb.	Au.	As.	Sb.	Au.	As.
	%			%			%			%		
1917	50	3.06	—	58	2.50	—	50	2.68	—	44	2.48	—
1920–21	57.86	2.01	.04	61.09	1.40	.03	45.27	3.32	3.07	50.31	1.32	1.33

Pure stibnite carries 71.4 % of antimony, so that in the highest-grade ore there are impurities whose nature is reflected in the average assays. In the hand-picked ore a higher gold content is associated with the lower percentage of antimony. This is not simply due to the variation in the reefs, and may be partly accounted for by the presence of more interstitial quartz in the lower grade stibnite if the interstitial quartz carries free gold. The coarse concentrates consist of particles varying in size from ⅛ to ½ in. in diam., and their percentage of antimony is uniformly higher than that of the hand-picked ore The percentage of interstitial quartz particles is therefore less, and the gold content is uniformly less than that of the hand-picked ore. Compared with the hand-picked ore and compared with each other, the coarse concentrates indicate that with less interstitial quartz there is less gold in the assay.

The impurities in the fine concentrates consist of pyrite and arsenopyrite, and the size of the particles in this product is so small that there is relatively much less included quartz than in the larger-sized products. The presence of the foreign sulphides accounts for the lower and more variable grade of antimony and for the percentage of arsenic in the fine concentrates. All the free gold in the crushed quartz is recovered in the fine concentrates, which consequently possess higher gold values. With the finer grinding it is noticeable that these gold values are much higher, and that the lower gold values in the hand-picked ore and the coarse concentrates are balanced by higher gold values in the fine concentrates. The finer grinding causes more of the stibnite to be carried as slime, and consequently the flotation concentrates contain a higher percentage of antimony, but at the same time the gold values are lower. As the finer grinding thus separates more of the quartz from the stibnite, so also more of the gold is separated and collected as free gold. It might be expected that if the size of the ore particles could be further reduced by grinding still more of the gold might be collected as free gold. The comparison of the values of the mill products, therefore, indicates that minute particles of free gold exist in the ore either embedded in the minute particles of quartz or in the stibnite, but it does not determine whether all the gold exists in this form.

3. *Quartz Reefs.*—Apart from the vertical veins carrying payable gold and antimony, there are quartz reefs occurring along fault-planes. These are fault reefs, and are more commonly associated with the oblique faults. Some are over 4 ft. wide, but, wherever intersected in the crosscuts, this type of reef is valueless. In places the quartz shows slight lamination, and in addition a brecciated structure produced by the separation of the fragments of fractured country-rock by the growth of the quartz crystals.

At the intersection of one of these reefs with the Costerfield reef at the south end of the 420-ft. level an enrichment in gold occurred. The Western reef, west of Plowright's reef at the 420-ft. level, is intersected by this type of reef lying along the plane of a strike-fault. The stibnite reef is displaced by this fault, and above the intersection the Western reef is a thin quartz vein which gradually widens, and in a distance of 10 ft. opens out into 6 in. of stibnite, with 2 in. of quartz on either side of it. Some of the quartz in the fault-plane between the extremities of the Western reef is highly cavernous, and the vugs are lined with masses of small quartz crystals in which are set fine particles of free gold. In view of the normal absence of gold in this type of reef, and the relatively shallow depth of this occurrence, it seems likely that this free gold has been derived from the stibnite vein by secondary processes and by downward circulating solutions.

Another type of unproductive vein is the laminated quartz vein. In most cases the laminated veins are parallel to the bedding, and in a south branch crosscut at the 780-ft. level a laminated vein 3 in. wide can be observed to follow the arch of the anticline. Occasionally laminæ are observed in quartz in the fault reefs and in the stibnite veins. In the latter, lenses of stibnite may develop between the laminæ, but the bedded laminated veins are uniformly unproductive. Examination of the laminated quartz under the microscope shows clearly that the laminæ possess similar character, to the laminæ in the Bendigo quartz reefs. The laminæ consist partly of residual mudstone and partly of dark matter in tortuous, fragmentary lines similar to the carbonaceous laminæ of Bendigo The laminæ are, therefore, considered to represent the residuals carbonaceous, and unreplaced material left embedded in the vein after the more or less complete replacement of the mudstone by the mineral solutions percolating along the bedding planes. White mica is also found embedded in the quartz, and probably represents the re-crystallization of the fine scaly mica dissolved from the mudstone. A carbonate—probably ankerite—is fairly abundant in some specimens; arsenopyrite and pyrite are not uncommon, and occasional crystals of zinc-blende occur. The same process of replacement may proceed from closely-spaced fractures across the bedding, in which case the vertical reefs may contain occasional laminæ. The processes of vein-growth and replacement may be simultaneous at Costerfield as at Bendigo, and the evidence is seen in the quartz veins, 8 in. wide, where the clear quartz is bounded by a few laminæ on both sides.

4. *Relation of the Bedded Laminated Reefs to the Stibnite Veins.*—In the south level on Kendall's reef several laminated quartz veins are successively intersected by Kendall's reef at an acute angle. The laminated veins are bedded, and pitch about 15° N. Where intersected they are on the western side of the anticline through which Kendall's reef is cutting vertically. The continuous character of the stibnite through the laminated quartz suggests that the laminated veins are cut through by the quartz stibnite veins during a later stage of mineralization.

In an example in the north drive off the west branch crosscut 570 ft. S. at the 786-ft. level, a small $\frac{1}{4}$-in. stibnite vein was observed to intersect the laminated quartz (Pl. III., fig. 2). In the illustration the beds have been disturbed by two faults, which appear to displace the laminated quartz so that the latter would appear at first to be earlier than the fault. Closer examination, however, shows that the laminated quartz, passes out into thin quartz veins, one of which is lying along the fault, and part of the laminated quartz itself is also on a fault-plane. It is therefore unlikely that any of the quartz is earlier than the faulting. It is more likely that the deposition of the clear quartz and the laminated quartz has been contemporaneous, as has been the case at Bendigo. A vein of white quartz and dark stibnite enters the bottom of the drive with the fault, but turns and leaves the fault as it butts against the laminated quartz. The stibnite and this white quartz are clearly younger than the faults. Examination of a thin section of the intersection of laminated quartz and a quartz-stibnite vein discloses the fact that particles of stibnite extend irregularly for short distances into the laminated vein. These associations of quartz and stibnite support the view that, whereas the deposition of quartz has accompanied the deposition of stibnite, the formation of the laminated veins has taken place in the earlier stages of the circulation of the mineral solutions that finally produced the stibnite veins.

5. *Formation of the Veins.*—The productive veins represent simple vein-growths from mineral solutions carrying antimony sulphide and silica, with small quantities of iron and arsenic sulphides and gold. These veins trend more or less at right-angles to the fault-planes, so that the separation of the vein walls by crystal growth has taken place more or less in the direction of the faults. The adjusting rock-movements, resulting from the widening growth of the vein, have thus taken place along the fault-planes which were the existing planes of weakness in the rock during the period of vein-formation.*

The main courses of circulation of the ascending solutions have been the pre-existing fault-planes and joint-planes in the region of the anticline. The small amount of mineralization in the wall-rock and the subordinate amount of laminated quartz in the veins (representing the extent of replacement) may be attributed to the dense character of the mudstone—*i.e.*, the porosity of the rock to the solutions has been very low. The material for the growth of the vein mineral has therefore been mainly supplied along the course of the veins from the solutions circulating along joints and faults.

The close association between quartz and stibnite is evidenced by the quartz selvage on the walls of the vein and by the distribution of the quartz particles through the massive stibnite. Some of the richest antimony veins, when traced along their strike, pass into quartz-stibnite veins, and from those into pure quartz veins. It is abundantly clear that the mineral solutions contained both stibnite and quartz.

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* "The Factors Influencing Gold Deposition in the Bendigo Goldfield," part 3. F. L. Stillwell. Add. Counc. Sci. and Ind. Bull. 16, 1919, p. 19.

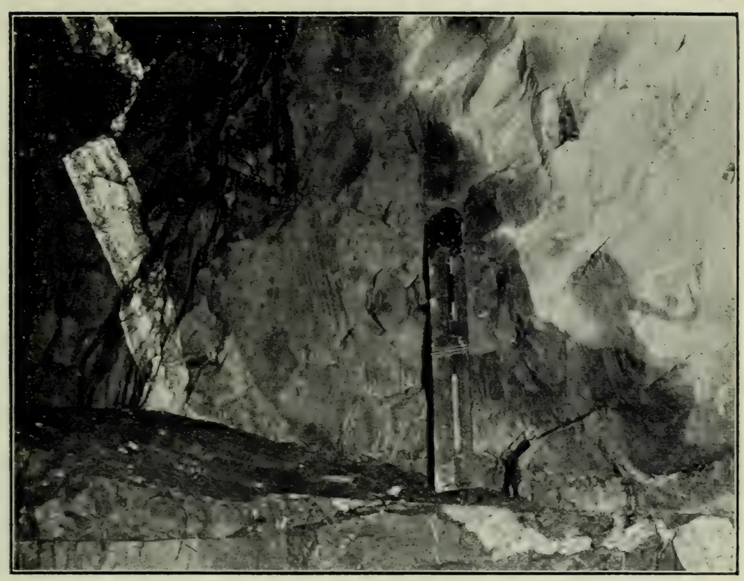

FIG 1.

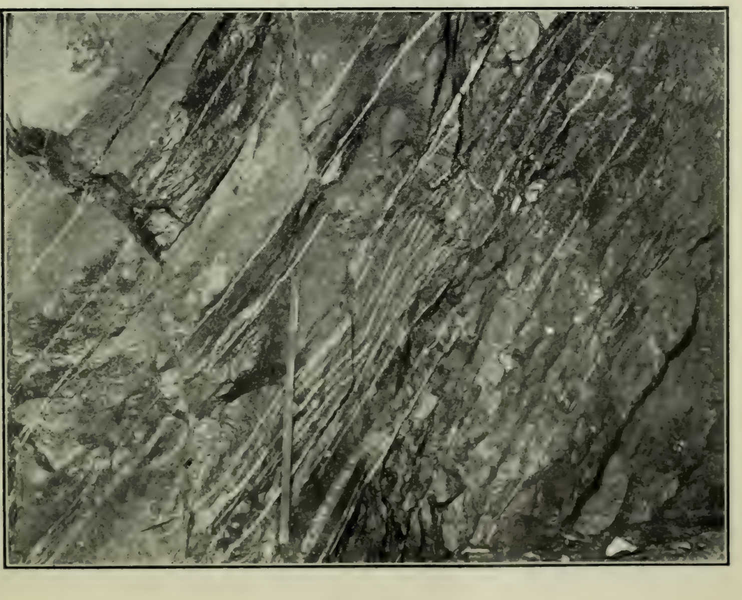

FIG. 2.

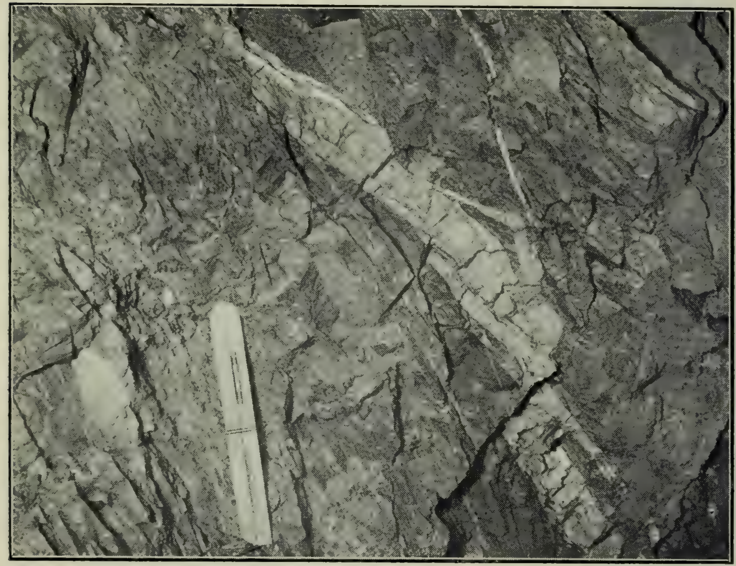

FIG. 1.

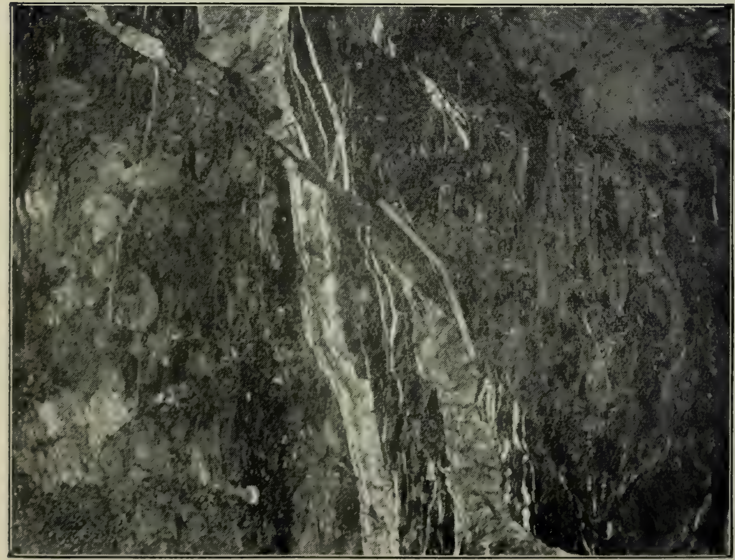

FIG. 2.

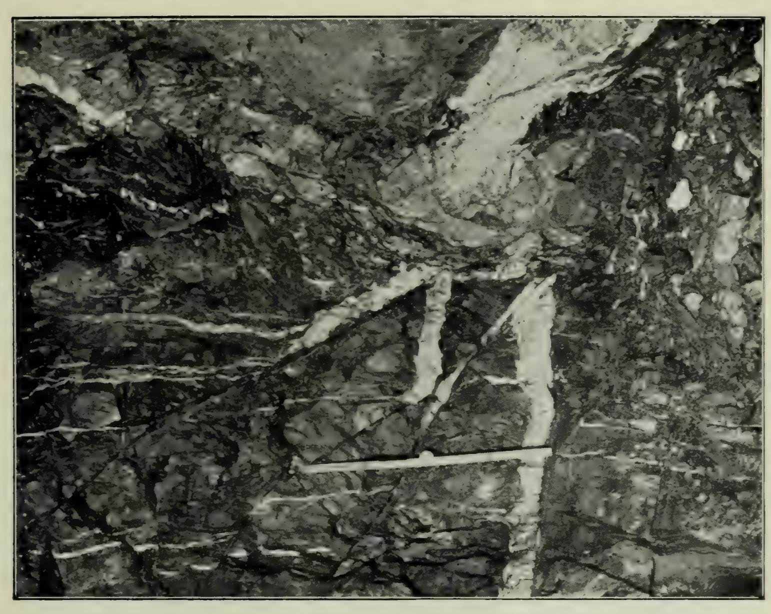

FIG. 2.

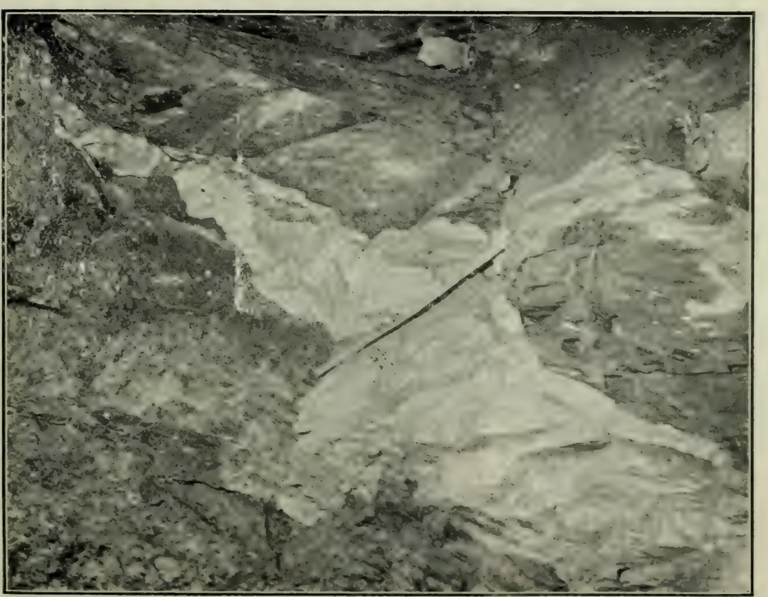

FIG. 1.

To a differential permeability of the vein walls may be attributed the migration of the iron sulphide, arsenopyrite, and carbonate, and the retention of the stibnite in the main vein. The separation is complete at the main Costerfield mine, where there are only small quantities of foreign sulphides.

There also appears to be a differential permeability between the quartz and stibnite along the course of the vein. Quartz threads are not uncommon in the mudstone adjacent to the stibnite veins, and, while these may unite with the main vein (Pl. II., fig. 1), the stibnite is confined to the main vein. This suggests that the quartz has crystallized in positions which have not been reached by the stibnite. The bulgy character (Fig. 3) is a feature of the stibnite veins which is absent when the vein contains a large percentage of quartz (Pl. I., fig. 1). The bulgy character would appear to be due to the slight changes or physical restrictions which have assisted or impeded the crystallization of the stibnite. These slight changes have not similarly affected the crystallization of the quartz, and in this respect there appears to have been a greater permeability of the quartz along the course of the vein.

The greater ease of penetration of quartz and the foreign sulphides is probably exemplified where these minerals have replaced the country-rock along the bedding-planes and produced the laminated quartz-veins without stibnite. It may also explain why some of the veins in the system illustrated in Pl. III., fig. 2, consist of quartz and others consist of quartz with a little stibnite, and others consist of stibnite with a little quartz.

The deposition of the stibnite has probably occurred in the veins by the direct crystallization from saturated solutions, and most of the gold has been precipitated with the sulphide. As the antimony veins increase in width they extend themselves in length, and the quartzy extensions of some of these veins may also be partly due either to the greater ease with which the siliceous matter is retained in the solutions or to the siliceous residuum after the crystallization of the bulk of antimony and gold. The crystallization of the quartz commenced with the formation of the laminated veins, continued through the development of the stibnite veins, and extended throughout the development of the barren fault reefs.

There may also have been a gradual change in the nature of the circulating solutions, which may have become mainly siliceous in the later stages. The final phase of mineralization probably produced the barren fault reefs.

It is possible that the primary solutions contained alkaline sulphides in which it is known that both stibnite and gold are soluble. It is also known that silver sulphide is practically insoluble in such solutions, and this fact may account for the absence of silver, except in traces, in the Costerfield gold. The ultimate source of these solutions is presumed to be differentiations from the igneous magmas which produced the granitic igneous masses outcropping to the south at Tooborac.

6. *Prospecting.*—The guiding principle in the development of the antimony veins is to follow the ore. Once the vein is intersected it is followed by rising, winzing, driving, and stoping in order to determine its extent and position. Branching veins of ore from the stopes are followed, and in some cases have become important producers. The want of persistency of an individual vein in strike and in height necessitates a very cautious policy in planning development work on the calculated position of any vein.

The resources of the main mine are considerable, and the known ore reserves have been increased by the results of important recent development work. Payable veins were followed as far as possible by winzes below the 786-ft. level before shaft-sinking was resumed in 1920. A sink of 200 ft. has been carried out, and the main crosscuts at the 885-ft. and 985-ft. levels are partly driven. There is no alteration in the nature of the country-rock in the bottom levels, nor any reason to expect any change in the conditions of ore-deposition in depth. A new payable vein has already been intersected in the crosscut at the 885-ft. level, and other veins can be confidently expected.

The immediate prospects are bright in the southern end of the mine, 1050 ft. south of the main shaft, at the 585-ft. and 686-ft. levels, where payable ore is being followed in the south drives. The possible scope of operations in this direction at these depths is enormous, as the ground southwards is virgin.

Prospecting by branch crosscuts from existing levels has been successful on two recent occasions. In the eastern ground, at the 585-ft. level, a branch crosscut has intersected the downward continuation of the old Costerfield reef, carrying payable values. This reef has been worked in the upper levels for an approximate length of 2000 ft., and was the main producer in the early days of the field. If it should maintain its values over a similar length at this depth the development will be one of great importance to the field. In the western ground, at the 420-ft. level, a branch west crosscut from Kendall's reef has intersected a new payable vein whose extension in height, depth, and strike will lead to the development of the western ground at both higher and lower levels in the neighbourhood of the shaft.

The discovery of a new reef such as this, which may extend for a considerable distance and be ultimately of great importance, illustrates the general method of development. Its extension in strike could now be tested by driving other branch crosscuts from Kendall's reef, and fresh points of attack possibly developed, from which the tonnage could be increased. The extension of the reef will, however, be proved or disproved by the more cautious policy of following the ore by driving levels along it and by rising and winzing. Experience of the uncertainty of the behaviour of the veins has proved the cautious method to be the most economical.

IV.—SUMMARY.

General.—Costerfield mining can be separated into three periods. In the first (and shortest) period of the "early sixties" high gold values were obtained in the oxidized zone. The second period of the "seventies" was occupied with the production of antimony ore, from which the gold recovery was very low. The third and present period commenced in 1906, and is distinguished by the complete recovery of the gold from the antimony and the elimination of losses in treatment. Within the third period a large tonnage of concentrates, equivalent to the whole of the Victorian production prior to 1906, has been produced, averaging 50.7 % antimony and 2.603 oz gold per ton.

The Country-Rock.—The country-rock is a dense Silurian mud stone, which is folded into a broad, low anticline. The mudstone is remarkably uniform to 1000 ft., the present depth of penetration. It is sometimes indurated, and this is mainly due to the development of the secondary minerals, pyrite, arsenopyrite, and carbonate.

Faults.—The main series of faults are a set of west-dipping faults which have a northerly strike and pitch. There are also a number of oblique faults. The reefs are younger than the faults.

Reefs.—The reefs consist of stibnite-veins and quartz-veins. The stibnite-veins are never free from quartz, and may grade into quartz-veins. Quartz-veins also occur which do not contain stibnite or gold, and are valueless.

The stibnite-bearing veins are vertical, simple lodes which form a branching system in the region of the anticline, both in plan and in transverse section. The veins average about 6 in. in width. The Costerfield reef, on the eastern side of this system, was the main reef in the early periods. Kendall's reef has been the main reef of the present period, but other reefs, known as Norris's, Teague's, Plowright's, "Eastern," "Far Eastern," and "Western" have been worked in the main mine.

Mineral Composition.—The mineral composition of the stibnite reefs is chiefly quartz, stibnite, and gold. The stibnite is remarkably free from arsenic and other impurities. The pyrite and arsenopyrite occur in small quantities, but are mainly confined to the wall-rock. The stibnite reefs may appear as massive veins of sulphide with thin selvages of quartz. In these cases they usually form dense, steely masses with high gold values. The prismatic crystal structure of stibnite is usually more noticeable as the percentage of quartz increases.

The gold is found as (1) free yellow gold, (2) free rusty gold, and (3) combined gold. The free yellow gold occurs embedded in the quartz. The free rusty gold occurs with the stibnite, and will not amalgamate with mercury, being presumably covered with a thin film of sulphide. Its colour is only skin deep, and it usually occurs near small vesicles with a spongy appearance, which suggests a secondary origin. In addition to the free gold, there is a considerable amount of gold, classed as combined gold, whose presence is only revealed by assay.

A comparison of two sets of average values of mill products shows that, with less interstitial quartz, there is less gold, and indicates that minute particles of free gold exist embedded in the stibnite or interstitial quartz. It does not seem probable that all the gold exists in this form.

Quartz Reefs.—Quartz reefs lie along some of the fault-planes, and laminated quartz-veins along some of the bedding-planes. These reefs are valueless wherever intersected. The laminated veins are usually about 3 in. wide, and the laminæ are considered to represent the residual, carbonaceous, and unreplaced material left embedded in the vein after the replacement, which has extended from the bedding plane or fracture. The process of vein-growth accompanies the process of replacement. The laminated veins are intersected by the stibnite veins, and are considered to have been formed in the earlier stages of the circulation of the mineral solutions.

Formation of Veins.—The stibnite veins represent simple vein growths from mineral solutions carrying antimony sulphide, silica, small quantities of iron and arsenic sulphides, and gold, which have percolated from pre-existing fault-planes and joint-planes along the course of the veins. The deposition of gold and antimony has probably occurred from the direct crystallization of saturated solutions.

The deposition of the antimony as sulphide appears to have been assisted by varying physical conditions which have not affected the quartz. The vein-walls appear to have been permeable to pyrite and arsenopyrite, and relatively impermeable to stibnite. The small quantities of these foreign sulphides at Costerfield are in this way naturally separated from the stibnite. The mineral solutions may have become more siliceous in the later stages of mineralization, when the barren fault reefs have been produced. The solutions are presumed to be differentiations from igneous granitic magmas.

Prospecting.—Owing to a characteristic want of persistency in strike of an individual antimony vein, the important principle in prospecting the veins is to follow the ore. In this way branching reefs are developed.

ACKNOWLEDGMENT.

In conclusion, the writer wishes to acknowledge his indebtedness to Mr. J. J. Doyle, J.P., general manager of the Costerfield Mining Company, whose kind assistance in every direction has made possible the compilation of these notes. The co-operation of the Victorian Geological Survey and the Geological Department of the Melbourne University is gratefully recorded.

Description of Plates.

Plate I., fig. 1.—A 3-in. vein of quartz and stibnite, apparently truncated by faults in the branch west crosscut, about 500 ft. S. of the shaft at the 686-ft. level, main shaft, Costerfield. The ends of the vein are connected by an irregular vein of quartz and antimony along the faults.

Plate I., fig. 2.—System of thin veins, some of which consist of quartz, some of quartz with a little stibnite, and some of stibnite. Near north end of intermediate drive on Teague's reef below the 686-ft. level, main shaft, Costerfield.

Plate II., fig. 1.—Stibnite vein (Teague's reef) in the south face of the intermediate level below the 686-ft. level, main shaft, Costerfield. Part of the vein shows a quartz selvage, and there is a thin branching quartz-vein which does not contain stibnite.

Plate II., fig. 2.—Face of the main south drive at the 686-ft. level, 1050 ft. south of the shaft, main mine, Costerfield. A massive stibnite vein and a quartz-stibnite vein are shown intersecting a fault.

Plate III., fig. 1.—Stibnite reef 12 in. wide on wall of Mason's drive, 500 ft. south of the shaft, 786-ft. level mine shaft, Costerfield.

Plate III., fig. 2.—Face of north drive from the western branch crosscut, 570 ft. S. of the shaft at the 786-ft. level, main shaft, Costerfield. Laminated quartz is disturbed by east-dipping faults (dotted). A stibnite vein (A) enters the drive with the fault, meets the laminated quartz, and then turns away from the fault.

Arnall & Jackson, Printers, Melbourne.

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PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED).

NEW SERIES.

No. 49.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

31st MARCH, 1923.

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MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

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Mount Morgan -	B. G. Patterson, Mount Morgan, Q.
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The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

Collins Court, 374 Little Collins Street, Melbourne, Victoria.

Institute Matters.

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MINUTES OF MEETINGS

OF THE INSTITUTE.

ANNUAL MEETING,

MELBOURNE, 26TH MARCH, 1923,

IN THE INSTITUTE ROOMS, 374 LITTLE COLLINS-STREET, AT 1 P.M.

Mr. Colin Fraser (President) occupied the chair.

The Secretary read the notice convening the meeting.

Minutes of the First Ordinary Meeting, 1922, were confirmed.

The Report of the Council and Balance-Sheet for 1922 were submitted and adopted.

Messrs. Oxlade and Mackie were elected Auditors for 1923 at a remuneration of £2 2s.

Report of Council for 1922.

To the Members.

The Council has pleasure in presenting its report for the year ended 31st December, 1922.

Membership of the Institute now totals 584, made up as follows:—Life Members, 6; Members, 361; Associate Members, 163; Juniors, 2 and Students, 52. Five Members, 7 Associate Members, 1 Junior and 8 Students were admitted during the year. Five Members and 1 Associate Member died. Ten Members, 2 Associate Members, and 1 Student resigned.

The following officers were elected to fill the vacancies caused by the annual retirements, in accordance with the rules:—

President: Colin Fraser.
Vice-President: H. W. Gepp.
Council: *Australasia*—E. G. Banks, J. M. Bridge, G. D. Delprat, and James Horsburgh.
New South Wales—G. H. Blakemore, H. A. Evans, and F. Voss Smith.
Victoria—Professor E. W. Skeats and G. H. Broome.
Queensland—A. A. Boyd.
Western Australia—Rd. Hamilton.
Tasmania—R. M. Murray.
Hon. Treasurer: A. S. Kenyon.

Committees as follow were appointed early in the year:—Executive, Publication and Education, Finance, Branches, and Mine Managers' Certificates.

It is with regret that the Council has to report the loss the Institute has sustained by the death of the following members:—Robert C. Sticht, C. T. Townsend, John W. Ashcroft, Alfred J. Higgin, and Gordon L. A. Ditchburn (Associate member). Mr. Sticht was a Past President of the Institute and a member of Council at the time of his death. He took a deep interest in general science, literature, and art, and was one of the most prominent metallurgists of the day. His knowledge of pyritic smelting was profound, and his contributions to the literature on that subject were of great value. He was a contributor to the publications of the Institute. Mr. A. J. Higgin was Lecturer on Metallurgy at the Melbourne University, and was on leave of absence in England at the time of his death. Mr. J. W. Ashcroft was a well-known consulting mining engineer in Sydney, and was admitted to the Institute twenty years ago. Mr. J. A. Hughes joined the staff of the Charters Towers School of Mines as Junior Demonstrator in Chemistry and Assaying; later he became Senior Demonstrator on that subject, and finally was promoted to the position of Lecturer in Chemistry, which position he held at the time of his death. Mr. C. T. Townsend specialized in the cyanide process at Charters Towers, and later at One-Mile, Gympie, Queensland. He joined the Institute in 1908. Mr. Gordon L. A. Ditchburn received his technical education at the Working Men's College, Melbourne, and subsequently gained experience on the Speakman's Gold Mines and the Ivanhoe Gold Corporation, Kalgoorlie; later he joined the staff of the Pahang Consolidated Co., Pahang, F.M.S., and at the time of his death occupied a responsible position on the staff of the Yukon Dredging Co., Kuala Lumpur, Federated Malay States.

Two meetings of the Institute were held during the year—the Annual Meeting in Melbourne on 27th March, and the First Ordinary Meeting at Port Pirie, South Australia, on 25th to 30th September. The First Ordinary Meeting included inspection of the works of the Broken Hill Associated Smelters at Port Pirie and of the Broken Hill Proprietary Co. at Whyalla and Iron Knob. The courtesy and assistance of these companies on that occasion is much appreciated. An account of the visit is recorded in Proceedings, No. 47. Papers prepared by members of the staff of the Broken Hill Associated Smelters also appear in that number, and are a valuable addition to metallurgical literature. A meeting of the full Council was convened to take place at Port Pirie coincident with the First Ordinary Meeting, but lapsed owing to want of a quorum. Members present were, however, invited to discuss Institute matters, and as a result the following recommendations to the Council were framed:—

(1) That the indexing of mining literature be again urged by the Council.

(2) That the Institute should be represented on all commissions, etc., dealing with the mining industry or any phase thereof.

(3) That the Council should do its utmost to further the proposed Federal Geological Survey as an aid to prospecting, and consider whether the Survey should specialize along oil-prospecting lines.

(4) That the Federal Engineering Council be requested to get into touch with the various authorities and urge that concessions in railway fares be granted to members of the Institute travelling from the various States to meetings,even if less than the stipulated six travel from any particular State.

(5) That the locality of the First Ordinary Meeting be left to the Council to decide.

During the visit to Port Pirie an informal meeting of local members and visitors was held to discuss the formation of a local branch, and some valuable suggestions on the subject were submitted by members of the Broken Hill Branch. It is anticipated that a Port Pirie Branch will be established during 1923.

The activities of the Broken Hill Branch are well maintained, and its work is much appreciated by the Council, not only on account of the valuable papers and discussion contributed by members, but by reason of the interest shown in matters affecting the mining industry generally.

The formation of branches in other centres is desired by the Council, and it is sincerely hoped that during the ensuing year earnest effort will be made to extend the scope and usefulness of the Institute by this means.

During the year Mr. W. E. Wainwright (member of Council) was appointed a member of the Engineering Faculty of the Melbourne University in place of Mr. G. C. Klug, who was visiting England.

Recognizing the value of geological surveys as an aid to prospecting, the Council is urging an extension of the survey of Broken Hill, recently completed by Mr. E. C. Andrews, Government Geologist, N.S.W. The area which it is suggested the survey should cover comprises an extensive tract of mineral-bearing country in the northwest of New South Wales.

The Council desires to record its thanks to Mr. C. F. Courtney, general manager of the Sulphide Coroporation Ltd., for a valuable donation of a set of Proceedings of the Institution of Civil Engineers and of Journals of the Royal Society of Arts.

Uniform standards of examination for mine managers' certificates and reciprocity between the various States has been urged by the Council for some time, and at the end of 1921 the Secretary for Mines, N.S.W., was requested to convene a conference of Chief Inspectors of Mines for the purpose of drafting a Bill to meet the case. This request has not yet been acceded to, but is still being pressed.

In April the Head Office of the Institute was removed to Collins Court, 374 Little Collins-street, Melbourne. The new quarters are modern, roomy, and well lighted, and are located in a suitable part of the city. Under the same roof are the Australian Mines and Metals Association, the Australian Metal Exchange, the Chamber of Mines of Victoria, and the Gold Producers' Association.

In August, 1921, the Australian National Research Council appointed a committee to consider the proposal to establish a Federal Geological Survey. The Aus. I.M.M. is represented on that committee. The Research Council urged that a Federal Geological Survey be established. Draft recommendations, to which members of the Council of the Aus. I.M.M. contributed, have been prepared and circulated, and are now under review.

An *Architects' Registration Act* (Victoria) was passed by Parliament during the year, and contains clauses protecting the rights of engineers. These clauses were recommended by the Council of this Institute and other engineering associations.

Recommendations were framed by the Council during the year with regard to misleading reports in prospectuses, and it was urged that legislation be enacted—

(1) To ensure that the prospectus of every new mining or metallurgical enterprise intended for public subscription be supported by a synopsis of the latest report (if any) by the State Geological Survey or Mines Department on the area, such synopsis to be previously approved by the department concerned.

(2) To render it obligatory that persons making technical reports for the promotion of mining companies should state in the report their degree or other qualifications, and should state the university or other institution from which they obtained such qualifications.

(3) To urge the immediate necessity for a mining companies law, especially relating to prospectuses, and providing that directors shall be personally liable for any wilfully misleading statements in such prospectuses.

The question of registration of professional engineers as proposed by the Institution of Engineers, Australia, is still under discussion by the Council. Opinions vary considerably as to the wisdom of applying registration to mining engineers. Those in favour of registration approve with reservations. Those against submit that, as the object of registration is to protect the public against employing men not qualified or capable to do certain work, and to secure work being entrusted to men capable of carrying out such work satisfactorily, this object will not be attained because—

(1) Where so many men were to be registered as professional engineers there must of necessity be amongst them some who are not entirely qualified or suitable to undertake special work which they might seek.

(2) An employer who intends to engage an engineer for certain work would not be satisfied merely with the statement that the applicant is a properly registered engineer. He would make inquiries as to his past experience and training, just as if no registration existed. The registration of engineers would not avoid this. Hence the employer would not be saved any trouble.

(3) One result of registration would be that many men who lack proper training would be excluded from obtaining employment. Practically they are so excluded now.

(4) There are men with a genius for engineering who have had no regular training, but who are capable of carrying out efficiently certain classes of engineering work.

(5) The result of registration would not give effective protection to the public, but would exclude a small number of men whose training is deficient and whose number is so limited that it is not worth while troubling about.

(6) The public would get a false sense of security should they rely upon registration being a guarantee of efficiency.

Federated American Engineering Societies, in a memorandum recently issued, submit "that the reasons which justify the registration of engineers responsible for public works do not apply to mining engineers." This is a subject for careful consideration by engineering institutions individually and by the Federal Engineering Council, Australia.

A conference of representatives of the Australasian Institute of Mining and Metallurgy, the Institution of Engineers, Australia, and the Australian Chemical Institute was held in Melbourne on 2nd February to further the proposal to establish a Federal Engineering Council. The resolutions of the previous conference held in Sydney on 22nd and 25th August, 1921, were reconsidered, and, with minor modifications, adopted for submission to the respective Councils for confirmation. These resolutions have since been approved, the constitution accepted by the three bodies, and "The Federal Engineering Council Australia" established. Draft by-laws under the constitution are now under consideration. Briefly, the objects of the Federal Engineering Council are to co-ordinate the activities of the constituent bodies and to consider and act upon all matters of common concern. The Aus. I.M.M. representatives on the Federal Council are Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power.

The establishment of an employment bureau for the use of members of the three institutes comprising the Federal Engineering Council was suggested by the Council of the Aus. I.M.M., and a scheme in this connection is being prepared by the Federal Council. Under the scheme proposed members seeking employment could refer, and persons, firms, etc., having vacancies could register.

The Council was represented at a conference held in Melbourne on 3rd February to consider the formation of an Engineering Standards Association. Several important recommendations were made and submitted to the Minister of Trade and Customs on 26th May. The recommendations were subsequently approved by the Federal Government, and a Commonwealth Engineering Standards Association finally established. The *Commonwealth Government Gazette* of 12th October contained the notification of appointments to the main committee. The bodies represented are :—The Institute of Science and Industry, State Governments, Institution of Engineers, Australia, the Australasian Institute of Mining and Metallurgy, and the Australian Chemical Institute. The functions of the Main Committee are :—

(*a*) To decide what standardization work shall be undertaken.
(*b*) To appoint members to sectional committees or panels to which the work of preparing specifications will be entrusted.
(*c*) To arrange for the carrying out of research work on the recommendations of the sectional committee.
(*d*) To receive and deal with the reports and recommendations of sectional committees.
(*e*) To arrange for the publication of the specifications through a standing committee.
(*f*) To keep in touch with "engineering standards" organizations in other countries.
(*g*) To control finance through a standing committee.
(*h*) To control the secretarial staff of the association.

The Australasian Institute of Mining and Metallurgy is represented on the main committee by Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power. The first meeting of the main committee was held in November.

The following were appointed delegates to the General Council of the Australasian Association for the Advancement of Science for the Wellington meeting, commencing on 9th January, 1923:—Professor Skeats, Professor Waters, and Messrs. G. D. Delprat, A. S. Kenyon, and E. G. Banks.

The Sixth International Mining Exhibition will be held in London from 1st June to 14th June, 1923, and, at the invitation of the directors of the exhibition, the patronage of the Aus. I.M.M. has been accorded. Members present in London on the dates mentioned are requested to pay the exhibition a visit.

The Council is pleased to report that, on the representation of the Australian National Research Council, the next Pan-Pacific Congress will be held in Australia. A cordial invitation is extended to members of the Aus. I.M.M. to take part in the congress, which opens in Melbourne on 13th August, 1923, and terminates there on 22nd August, when members will leave for Sydney. The Sydney meeting will open on 23rd August and terminate on 3rd September. A strong committee, on which the names of three members of the Aus. I.M.M. appear, has charge of affairs. The suggestion has been forwarded to the committee by your Council that Broken Hill be included in the itinerary of the congress.

The President of the Institute for 1921, Mr. G. C. Klug, visited London during the year, and whilst there revived the question of affiliation with the Institution of Mining and Metallurgy. Details of a scheme which provides for the affiliation of kindred institutions throughout the Empire have been prepared, and are expected to arrive in the near future.

The opinion of the Council was sought during the year in regard to the Chemical Engineering diploma course and entrance requirements of the Sydney Technical College. The recommendations of the Council thereon were forwarded to the local committee in Sydney, and eventually submitted by that committee to the technical staff of the College. Briefly, the opinions submitted were as follows :—

(1) That, though a foreign language is desirable, it would be a mistake to oblige diploma students to take one up as a part of their course.

(2) That the first three years of the course remain as at present.

(3) That Applied Mechanics, III., and Heat Engines, II., be deleted, and Industrial Chemistry and Non-Ferrous Metallurgy substituted.

(4) That Engineering Drawing and Design, II., be altered to Metallurgical Drawing and Design, which will also include Building Construction.

(5) That a short course of Business Methods be included.

The number and quality of papers published during the year is a matter upon which members may be congratulated, and the Council wishes here to record its thanks to the authors of these valuable papers for the care and trouble taken in their preparation, also to those who contributed discussion. In this matter alone the Institute serves a great purpose.

The success of the mining industry depends more and more each year on the advancement of the science and practice of mining and metallurgy, and every paper contributed helps to fulfil the duty entrusted to the Institute in this respect.

AUSTRALASIAN INSTITUTE OF MINING AND METALLURGY (Inc.).

Balance Sheet, 31st December, 1922.

LIABILITIES.

Subscriptions paid in advance				£11	11	0
Donation Trust A/c ...	£160	7	9			
Add Interest to 31st December, 1922 ...	8	0	0			
				168	7	9
Life Membership				36	15	0
Sundry Creditors				22	13	5
Accumulated Fund—						
Balance at 31st Dec., 1921	1818	0	0			
Less Balance transferred from Revenue A/c ...	23	13	9			
				1794	6	3
				£2033	13	5

ASSETS.

State Savings Bank at Credit	£55	1	6			
Commonwealth Bank at Credit	118	4	8			
				£173	6	2
Sundry Debtors				5	0	0
Commonwealth of Australia Inscribed Stock (at cost)	1439	7	6			
Commonwealth of Australia War Savings Certificates (at face value)	96	0	0			
				1535	7	6
Outstanding Subscriptions				155	0	6
Office Furniture, Melbourne and Broken Hill Branch				164	19	3
				£2033	13	5

Melbourne, 28th February, 1923.

Audited and found correct.

Oxlade & Mackie, Auditors.

STATEMENT OF RECEIPTS AND EXPENDITURE FOR THE YEAR ENDED 31ST DECEMBER, 1922.

RECEIPTS.

Dec. 31st, 1921.						
To Commonwealth Bank at Credit	£22	2	8			
State Savings Bank at Credit	100	0	0			
				£122	2	8
Dec. 31st, 1922.						
Members' Subs., 1920 ...	5	16	6			
" " 1921 ...	80	1	6			
" " 1922 ...	741	9	4			
" " 1923 ...	8	8	0			
Entrance Fees	23	13	3			
Advertisements ...	37	0	0			
Sales	11	6	11			
A. T. Danks Prize ...	10	0	0			
Refund-Mt. Morgan Branch	3	3	2			
Refund-Port Pirie Branch	14	13	9			
Refund—Letter of Credit, London	25	0	0			
Zinc Producers Association—Contra ...	11	4	3			
Interest--Commonwealth Bank	3	2	11			
Interest—State Savings Bank	5	1	6			
Interest—Inscribed Stock	96	0	0			
Sale of War Savings Certificates	251	18	0			
				1327	19	1
				£1450	1	9

EXPENDITURE.

Dec., 31st, 1922.						
By Printing & Stationery ...	£21	5	3			
Rent	50	0	0			
Salaries	273	0	0			
Postages, Telegrams, &c.	63	15	6			
Sundries	22	19	9			
Proceedings	547	14	6			
Meeting Expenses ...	50	0	0			
Commission	4	8	6			
Zinc Producers Association—Contra ...	11	4	3			
A. T. Danks Prize ...	9	12	6			
Law Costs	2	7	0			
Furniture	121	8	4			
War Savings Certificates at face value	96	0	0			
				£1276	15	7
Commonwealth Bank at Credit	118	4	8			
State Savings Bank at Credit	55	1	6			
				173	6	2
				£1450	1	9

Revenue Account for the Year ended 31st December, 1922.

1922.				1922.			
Dec. 31.	To Publications	£461 12 7		Dec. 31.	By Subs., 1920 ...	£5 16 6	
	Stationery & Printing ...	24 5 3			,, 1921 ...	80 1 6	
	Postages, Telegrams, &c....	63 15 6			,, 1922 ...	678 2 10	
	Salaries ...	273 0 0			Advertisements	42 0 0	
	Rent ...	50 0 0			Sales ...	11 6 11	
	Sundries ...	22 12 3			Entrance Fees	23 13 3	
	Commission	4 8 6			Interest ...	104 4 5	
	Meeting Expenses ...	32 3 1					£945 5 5
	Loss on realization of War Savings Certificates	29 2 0			Balance transferred to Accumulated Fund ...		23 13 9
	Interest on Donation Trust A/c...	8 0 0					
			£968 19 2				
			£968 19 2				£968 19 2

Melbourne, 28th February, 1923. Audited and found correct. Oxlade & Mackie, *Auditors.*

OF THE EXECUTIVE COMMITTEE.

(Summary.)

JANUARY 15TH, 1923—1 P.M.

The Secretary's report was presented and accounts to the amount of £54, including petty cash (January) £3, petty cash advance (1923) £10, and printing and stationery £36, were passed for payment.

The following were declared elected to the Institute (as Members) by ballot of the Council :—Alfred Henry P. Lowry and Frederick William Reid.

Nominations of the following were approved for ballot :—As Member.—Alfred Thomas Bennetts. As Associate Member.—Walter Henry James Cropp.

Resignations of Messrs. J. B. Scott, R. M'I. Moore, and L. G. Klem were accepted.

Messrs. W. E. Wainwright and A. S. Kenyon were appointed to represent the Institute at a conference to be convened for the purpose of nominating a representative to the Architects' Registration Board, Victoria.

Mr. H. W. Gepp was elected a Vice-President of the Institute as provided by Article 39, and Mr. A. S. Kenyon, Hon. Treasurer, as provided by Article 56.

Draft of proposed by-laws of the Federal Engineering Council, also proposed re-draft of Clauses 6, 7, and 8 of the Constitution, were submitted, and consideration deferred.

An invitation to members to attend the meeting of the Institution of Engineers, to be held in Adelaide in March, was received, and the Secretary's action in notifying members thereof was approved.

A report from Mr. F. Danvers Power of a conference between representatives of the Institute and members of the Sydney Technical College staff in the matter of the metallurgical course of the College was submitted, and recommendations approved.

A draft memorandum on the recommendations of the committee appointed by the Australian National Research Council was submitted, and consideration deferred.

A letter was submitted from the Victorian chairman of the Pan-Pacific Congress, reporting that the Institute's suggestion that

Broken Hill should be included in the itinerary of the Pan-Pacific Congress would be considered when the programme was being arranged, probably in March.

Particulars of an offer by the Rand Mutual Assurance Company of prizes for design or designs of the most practicable device for recording shaft signals were submitted. Decided that members be notified thereof.

FEBRUARY 23RD, 1923—1 P.M.

The Secretary's report was presented, and accounts to the amount of £15, including petty cash £9, and process blocks £6, were passed for payment.

The following were admitted to the Institute by ballot of members of Council:—Albert Thomas Bennetts (as Member) and Walter Henry James Cropp (as Associate Member).

Nominations of Messrs. Walter G. A. Smith and E. F. Harris were deferred for further inquiry.

Resignation of Mr. W. C. Davies was accepted.

The following members were nominated to sectional committees in connection with the Australian Engineering Standards Association:—Paints, etc.—Messrs. Gilbert Rigg and W. Baragwanath; Standard Methods of Testing—Messrs. David Avery and Gilbert Rigg; Electrical Standardization—Mr. F. J. Mars.

Resolved that copies of letter received from the Australian National Research Council *re* Pan-Pacific Congress be forwarded to all members of Council of the Aus. I.M.M., and particulars be incorporated (to members) in the next circular issued.

Suggestions from members of Council in regard to by-laws of the Federal Engineering Council were submitted, and directions given in regard to their despatch to the Federal Council. Certain slight alterations to the constitution of the Federal Engineering Council were approved.

It was reported that Mr. H. E. Morton had been nominated to represent engineering interests on the Board constituted under the provisions of the *Architects Registration Act* (Vic.)

Other routine business was conducted.

MARCH 12TH, 1923—1 P.M. (SPECIAL MEETING).

Receipts and expenditure for recent years were reviewed by the Hon. Treasurer, details of estimated expenditure for 1923 were discussed, and the following recommendations adopted :—

(1) That a bonus of £50 for 1922 be paid to the Secretary, and that his salary be increased by £100 per annum.

(2) That in regard to expenditure for 1923, the following be adopted as an estimate :—Printing, stationery, etc., £40 ; rent, £50 ; salaries, £376 ; postage, £80 ; sundries, £40 ; Proceedings, £500 ; commission, £7 ; meeting expenses, £60 ; Branches, £25 ; Engineering Standardization, £25 ; and contingencies, £50.

(3) That outstanding accounts on 31st December, 1922, be paid out of accumulated funds.

(4) That mining companies, selected by the Council, be written to by the Council for financial assistance towards publications.

(5) That it is desirable to increase the amount of members' subscription to £3 3s. per annum ; that if the Council favours the suggestion it take steps to submit same to the First Ordinary Meeting, 1923, before remission of the question to members for ballot in accordance with the Articles of Association.

(6) That mining companies, through the medium of the Australian Mines and Metals Association, be requested to pay the balance (£75) of the amount (£100) to which the Institute was committed in connection with the inauguration expenses of the Commonwealth Engineering Standards Association.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holiays.

MEMBERSHIP.

Since the last issue of Proceedings the following have been admitted to the Institute :—

Members.

Names and Addresses.

Bennetts, Albert Thomas, Bentong, Pahang, Federated Malay States.

Lowry, Alfred Henry Piper, c/o Broken Hill North Ltd., Broken Hill, N.S.W.

Reid, Frederick William, School of Mines, Adelaide, S.A.

Associate Member.

Cropp, Walter Henry James, c/o Mount Bischoff Tin Mining Co., Waratah, Tas.

FEDERAL ENGINEERING COUNCIL.

A draft of the proposed by-laws of the Federal Engineering Council has received the consideration of the Council of the Aus. I.M.M., and suggestions thereon have been forwarded to the head office in Sydney.

ARCHITECTS REGISTRATION ACT (VIC.)

Mr. H. E. Morton, of Melbourne, has been appointed to represent engineering interests on the Board constituted under the provisions of the *Architects Registration Act*, Victoria.

PAN-PACIFIC CONGRESS.

Members are reminded that the Pan-Pacific Congress will be held in Australia, commencing in August next. Particulars regarding membership of the Congress may be obtained from the hon. secretary, Australian National Research Council (Mr. R. H. Cambage), 5 Elizabeth-street, Sydney. It is hoped that the itinerary of the Congress will include a visit to Broken Hill via Cobar, returning by way of Iron Knob, Port Pirie, Adelaide, and Melbourne.

COMMONWEALTH ENGINEERING STANDARDS ASSOCIATION.

The second meeting of the main committee of the Australian Commonwealth Engineering Standards Association was held in

Sydney on 27th February, when a number of important matters, particularly the appointment of sectional committees, were considered. The Aus. I.M.M. is responsible for the sum of £100 as a contribution towards the preliminary expenses of the Association. Of this amount, certain mining companies have contributed £50.

MOVEMENTS OF MEMBERS.

W. H. C. **Chapple** is now residing at 49 Smith-street, Manly, N.S.W.

R. A. **Clinton** has left Iron Knob, where for some time he was on the staff of the Broken Hill Proprietary Co. Ltd.

W. H. **Cropp**, of Wellington, N.Z., is now assayer to the Mount Bischoff Tin Mining Co., Waratah, Tas.

H. **Charlton Dod**, of the S.A. Oil Wells Co., is at present located in Victoria. His address is Evans-street, Newtown, Geelong.

Edward C. **Dyason**, of Melbourne, is at present touring America.

T. J. **Dyson**, who has been for some years in London, returned to Australia early in Feburary, and is residing at Kilsyth, Vic.

Colin Fraser (President) sailed for London on 31st March on a holiday tour.

E. H. **Fraser** is now with the Electrolytic Zinc Co. at Zeehan, Tas.

James Hebbard, manager of the Central Mine, Broken Hill, is at present visiting Japan, and will be returning to Australia in May.

Jas. Hill is with the Main Roads Board, Cairns, Queensland.

F. C. **Johns**, late of the Junction North Mining Co., Broken Hill, has joined the staff of Messrs. White and Hosier, of that town.

A. J. **King**, of Federated Malay States, will shortly visit Australia.

G. C. **Klug** will leave Australia for London on 26th May. He expects to visit Australia occasionally.

W. G. **Langford**, manager in Papua for the Anglo-Persian Oil Company and Commonwealth representative on the Papuan Oil Fields, will be in Melbourne in April and May on leave.

A. D. **Mackay** is managing the Wombat Mine at Waratah, Tas.

M. R. **M'Keown** has been appointed manager of the Valla Gold Mines Ltd., Urunga, N.S.W.

E. N. **Mulligan** is on the staff of the Austral-Malay Tin Ltd., at Taiping, Federated Malay States.

J. R. **Peberdy** is leaving Australia early in May to join the staff of the Rawang Tin Ltd., near Kuala Lumpur, F.M.S.

BRANCH REPORTS.

The Annual General Meeting of the Broken Hill Branch was held on Tuesday evening, 27th March, 1923, when the following officers were elected for the ensuing session :—

Chairman—F. VOSS SMITH.

Vice-Chairmen—R. T. SLEE and E. B. DOW.

Secretary—C. H. J. CLAYTON.

Treasurer—P. H. WARREN.

Auditor—C. A. CHERRY.

Committee—A. S. CLARK, H. J. HARRISON, H. V. SEALE, R. D. NEVETT, E. T. HENDERSON, F. A. ROBERG, N. J. ROBERTS, H. H. CARROLL, F. N. THOMAS, W. B. CHOMLEY, and J. F. SHAW.

By permission of the Mine Managers' Association, a report to that body by Mr. G. L. Shirkill, dealing with mining drill-steel problems, was read and discussed.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

FROM 31ST DECEMBER, 1922, TO 31ST MARCH, 1923.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Queensland Department of Mines : Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines : Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines : Report of Executive	monthly	Bulawayo
State Electricity Commission of Victoria : Third Annual Report Short Description of Works at Yallourn		Melbourne
Institution of Engineers, Australia : Transactions, Vol. I.		Sydney
Royal Society of Queensland : Proceedings, Vol. XXXIV.		Brisbane
Queensland Department of Mines : Publication, No. 273		Brisbane
South Australian Department of Mines : Review of Mining Operations, No. 36		Adelaide

Royal Society of Tasmania :	
Papers and Proceedings, 1922	Hobart
New Zealand Board of Science and Art :	
Journal, Vol. V., No. 5	Wellington
Institution of Mining and Metallurgy :	
Bulletins, Nos. 220-222	London
Institution of Mining Engineers :	
Transactions, Vol. LXIV., Parts 3 and 5	London
Geological Society :	
Quarterly Journal, No. 312	London
Iron and Steel Institute :	
Journal, Vol. CVI., No. 2	London
Institute of Chemistry of Great Britain and Ireland :	
Proceedings, Parts 5 and 6, 1922 ; Part 1, 1923	London
Yorkshire Geological and Polytechnic Society :	
Proceedings, Vol. VI.	Leeds
Department of Scientific and Industrial Research :	
Fourth Report of Colloid Chemistry and its General and Industrial Applications	London
Royal Society of Arts :	
Journal, Vol. 70 (per Mr. C. F. Courtney)	London
Société des Ingeneiurs Civils de France :	
Bulletins, October to December, 1922	Paris
Annales des Mines :	
1922, Part 12; 1923, Parts 1 and 2	Paris
Mining and Geological Institute of India :	
Transactions, Vol. XVII., Parts 1 and 2	Calcutta
Department of Mines and Geology, Mysore :	
Records, Vol. XVIII., 1919, Part II; Vol. XIX., 1920, Part II ; Vol. XX, 1921, Part I	Bangalore
Geological Survey of India :	
Records, Vol. LIII., Part 4	Calcutta
Department of Mines, Canada :	
Summary Report, 1921, Parts B and E; Report for year ending 31st March, 1922	Ottawa
American Institute of Mining & Metallurgical Engineers :	
"Mining and Metallurgy" (monthly)	
Various Papers	New York

California State Mining Bureau:
  "Mining in California" (monthly)
  "Summary of Operations, California Oil Wells," (monthly) - - - - San Francisco
Secretaria de Industria Y Comercio:
  Bulletins, XIII., No. 6; Vol. XIV., No. 1 - Mexico
El Museo Nacional:
  Relatorio, 1922 - - - - Buenos Aires

RECENT ARTICLES ON MINING MATTERS.

(30th September,, 1922 to 31st December, 1922.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Engineering and Mining Journal Press.*

(6) *Annales des Mines*, Paris, France, mth.

(7) *Publications*, Department of Mines, Melbourne, Victoria.

(8) *Publications*, Department of Mines, Sydney, New South Wales.

(9) *Publications*, Department of Mines, Adelaide, South Australia.

(10) *Publications*, Department of Mines, Brisbane, Queensland.

(11) *Publications*, Department of Mines, Perth, Western Australia.

(12) *Publications*, Department of Mines, Hobart, Tasmania.

(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(14) *Publications*, Bureau of Mines, Toronto, Ontario.

(15) *Publications*, Geological Survey of India, Calcutta.

(16) *Publications*, Geological Survey, U.S.A., Washington.

(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(18) *Publications*, California State Mining Bureau, Sacramento, Cal.

(19) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.

(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(21) *Quarterly Journal*, Geological Society, London.

(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.

(23) *Transactions*, Inst. Min. Eng., London.

(24) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(26) *Transactions*, Am. Inst. of Min. & Met. Eng., New York City.

(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(28) *Journal*, Franklin Inst., Philadelphia, Pa.

(29) *Chemical Engineering and Mining Review.*

(30) *Transactions*, Am. Soc. C.E., New York City.

(31) *Bulletins*, Société des Ingénieurs Civils, Paris.

(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(33) *Publications*, Iron and Steel Institute, London.

(34) *Proceedings*, Inst. of Mech. Eng., London.

(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(37) *Transactions*, Mining and Geological Institute of India, Calcutta.

(38) *Publications*, Department of Mines, Wellington, N.Z.

(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

LIST OF ARTICLES

GEOLOGY.

Origin, Migration, and Accumulation of Petroleum. R. E. Somers. (5) Jan. 13, 1923.

Geology of Prospecting. W. H. Emmons. (5) Jan. 20, 1923.

The Designation of Polymetallic Ore Deposits with Flag Marks. A. K. Boldirey. (5) March 3, 1923.

Economic Application of Zonal Theory of Primary Deposition of Ores. J. C. Anderson. (26) Jan., 1923.

Magnetic Methods for Exploration and Geologic Work. W. O. Hotchkiss. (26) Feb., 1923.

Some Notes on the Secondary Enrichment Exhibited by Certain Auriferous Veins: Discussion. P. C. Whitehead. (22) Bull., Jan., 1923.

MECHANICAL.

Advantages of Basic-Electric-Furnace Iron for Castings to Resist Abrasion. L. J. Barton. (5) April 7, 1923.

The Philosophy of Wire Rope. H. M. Hall. (5) April 7, 1923.

Review of Present Status of Drill Steel Breakage and Heat Treatment. F. B. Foley, C. Y. Clayton, and H. S. Burnholz. (26) Jan., 1923.

Hardness and Heat Treatment of Mining Drill Steel Shanks. C. Y. Clayton. (26) Jan., 1923.

Heating and Cooling Curves of Large Ingots. F. E. Bash. (26) Jan., 1923.

Design and Operation of Roberts Coke Oven. M. W. Ditto. (26) March, 1923.

Heat Distribution in New Type Koppers Coke Oven. J. Van Ackeren. (26) March, 1923.

Alcohol Fuel Mixtures. H. R. Adam. (25) Dec., 1922.

Combustion of Powdered Coal. H. Kreisinger and J. Blizard. (39) March, 1923.

METALLURGICAL.

Concentration of Rossland, B.C., Ores. D. Lay. (5) Dec. 21, 1922.

The Fiechtl Vertical Retort Furnace for Zinc Ores. W. Kapellmann. (5) Dec. 30, 1922.

Renewable Nose for Crusher Feed Spout. W. J. Tait. (5) Dec. 30, 1922.

Arc Regulation in Electric Furnaces and Pilot-Light Control. (5) Dec. 30, 1922.

Two Electrolytic Copper Processes Compared. A. G. Betts. (5) Jan. 6, 1923.

Possibilities of Dry Chlorination of Oxidized Zinc Materials. C. G. Maier. (5) Jan. 13, 1923.

Amalgamation, All-Cyanidation, and Thievery. A. W. Allen. (5) Jan. 13, 1923.

Milling and Flotation. A. F. Taggart. (5) Jan. 20, 1923.

Hydrometallurgy. R. Gahl. (5) Jan. 20, 1923.

Pyrometallurgy. E. H. Robie. (5) Jan. 20, 1923.

Mill Design and Construction. E. Gayford. (5) Jan. 20, 1923.

Refining Gold Bullion with Chlorine at the Ottawa Mint. A. H. W. Cleave and P. W. Bond. (5) Feb. 3, 1923.

Continuous *v.* Intermittent Smelting of Complex Zinc Ores. W. McA. Johnson. (5) Feb. 10, 1923.

The Rise and Fall of Metallurgical Processes. E. H. Robie. (5) Feb. 10, 1923.

Improvement in Crusher Steel is Suggested by Metallographic Tests. O. E. Harder. (5) Feb. 17, 1923.

The New Concentrator of the Real Compania Asturiana de Minas, Spain. L. Barcena y Diaz. (5) March 3, 1923.

Smelting Silica Bricks with Reverberatory Slags in a Copper Blast Furnace. R. F. Wood. (5) March 3, 1923.

Pyritic Smelting and Basic Converting at the Kosaka Copper Smelter, Japan. K. Ikeda. (5) March 3, 1923.

Improved Ball-Mill Screen. W. I. Nelson. (5) March 3, 1923.

The Use of Sodium Peroxide in Cyanidation at the Teck-Hughes Gold Mine. D. L. H. Forbes. (5) March 3, 1923.

Notes on East Butte's Mining Operations. F. E. Wormser. (5) March 10, 1923.

The Laws of Crushing. J. Herman. (5) March 17, 1923.

The Cyanide Plant of the Nurupi Mines, Korea. C. Flury. (5) March 31, 1923.

Combustion of Coke in Blast Furnace Hearth. G. St. J. Perrott and S. P. Kinney. (26) March, 1923.

Relative Efficiency of Amalgamation and Cyanidation. A. J. Clark and W. J. Sharwood. (26) March, 1922.

Heap Leaching at Bisbee. A. W. Hudson and G. D. Van Arsdale. (26) Feb., 1923.

Forms of Sulphur in Coke, and Their Relation to Blast Furnace Reactions. A. R. Powell. (26) Feb., 1923.

Note on Apparatus for Small-Scale Flotation Tests. H. L. Sulman and H. F. K. Picard. (22) Bull., Jan., 1923.

Recovery of Gold by Blanket Concentration. F. Wartenweiler. (25) Feb., 1923.

Determination of Small Quantities of Molybdenum in Tungsten. W. J. King. (39) April 1, 1923.

MINING.

Mining by Briggs' Underhand Square Setting. R. H. Dickson. (5) Jan. 6, 1923.

Mining Practice. G. J. Young. (5) Jan. 20, 1923.

Mining by Ratterree Modification of Underhand Square Setting. R. H. Dickson. (5) Jan. 27, 1923.

Inspiration Consolidated's Porphyry Shaft. A. C. Stoddard. (5) Feb. 10, 1923.

Porcupine: Premier Gold District of Northern Ontario. R. E. Hore. (5) Feb. 24, 1923.

Hydraulic Power *v.* Compressed Air as Applied to Rock Drills. B. V. Stuart Davies. (5) March 3, 1923.

Safety Guides for Inclined Shafts. E. D. Gardner. (5) March 17, 1923.

Mining an Upper Bituminous Seam After a Lower One has been Extracted. H. N. Eavenson (26) March, 1923.

Mining Methods at the Bunker Hill and Sullivan. H. M. Childs and S. A. Easton. (26) March, 1923.

Mining Methods of Hecla Mining Co. J. F. M'Carthy and C. H. Foreman. (26) March, 1923.

Mining Methods of the Copper Range Co., Houghton County, Mich. W. H. Schacht. (26) March, 1923.

Mining Methods of Marquette District, Michigan. S. R. Elliott, J. E. Jopling, R. J. Chenneour, and E. L. Derby. (26) March, 1923.

Liquid Oxygen Explosives at Pachuca. M. H. Kuryla and G. H. Clevenger. (26) March, 1923.

Liquid Oxygen in Mining. A. James. (22) Bull., Feb., 1923.

The Lowest Temperature Yet Attained. H. H. Paine. (25) Jan., 1923.

MISCELLANEOUS.

Marketing of Metalliferous Ores and Concentrates. A. B. Parsons. (5) Serial commenced Dec. 23, 1922.

Standardization of Tool Joints. S. A. Guiberson. (5) Jan. 6, 1923.

The Marketing of Mica. O. Bowles. (5) Jan. 13, 1923.

Standardization. C. A. Mitke. (5) Jan. 20, 1923.

Mining and Washing Phosphate Rock in Tennessee. R. W. Smith. (5) Feb. 3, 1923.

The Marketing of Bismuth. H. M. Brush. (5) Feb. 10, 1923.

Discussion of Measures for Prevention and Fighting Fires in Metal Mine Workings. C. A. Mitke. (5) Feb. 17, 1923.

Barytes: Occurrence, Mining, Uses. R. B. Latoo. (5) Feb. 17, 1923.

Tunnel Surveying at Potreillos, Chile. J. E. Harding. (5) Feb. 24, 1923.

Marketing of Chromite. E. Sampson. (5) Feb. 24, 1923.

Marketing of Marble. J. S. Sewell. (5) March 10, 1923.

Examination of Tin Deposits in the Federated Malay States. J. B. Newsom. (5) March 17, 1923.

Utilization of Waste Material. A. W. Allen. (5) March 17, 1923.

Alunite and Its Products. R. H. Tingley. (5) March 17, 1923.

Oil-Field Sampling. E. R. Lilley. (5) March 31, 1923.

Underground Orientation by Three-Wire Method. J. R. Jahn. (5) March 31, 1923.

Occurrence, Chemistry, and Uses of Selenium and Tellurium. V. Lenher. (26) Jan., 1923.

Relation of Bonuses and Costs to Present-day Prices of Crude and Its Products. T. Cox. (26) Jan., 1923.

Mine Drainage, Stream Pollution. A. B. Crichton. (26) March, 1923.

The Value of Rand Gold-Mining Investments. W. A. Caldecott and G. J. V. Clarence. (25) Journal, Dec., 1922.

Collection and Examination of Explosive Dusts in Air. L. J. Trostel and H. W. Frevert. (39) March, 1923.

The Manufacture of Sulphuric Acid. L. A. Pratt. (39) Jan., 1923.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED).

NEW SERIES.

No. 50.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

30th JUNE, 1923.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

LIST OF OFFICERS, 1923.

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A. S. KENYON.

Bankers:

COMMONWEALTH BANK OF AUSTRALIA, Collins Street, Melbourne, Vic.

Secretary:

D. L. STIRLING, Collins Court, 374 Little Collins Street, Melbourne, Vic.

BRANCH SECRETARIES:

Broken Hill - C. H. J. CLAYTON, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie - S. G. TURRELL, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan - B. G. PATTERSON, Mount Morgan, Q.
Port Pirie - G. DEY, c/o Broken Hill Associated Smelters, Port Pirie.
Sydney - F. DANVERS POWER, 31 Hunter Street, Sydney, N.S.W.

LOCAL CORRESPONDENTS:

South Australia - H. W. GARTRELL, The University, Adelaide.
New Zealand - D. B. WATERS, The University, Dunedin.
Tasmania - RUSSELL M. MURRAY, Queenstown.
North Queensland MURRAY RUSSELL, Department of Mines, Brisbane.
Papua - - W. G. LANGFORD, c/o Anglo-Persian Oil Co., Popo, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

COLLINS COURT, 374 LITTLE COLLINS STREET, MELBOURNE, VICTORIA.

Institute Matters.

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MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

Summary.

APRIL 30TH, 1923—1 P.M.

The Secretary's report was presented, and accounts to the amount of £73 10s. 7d., including salary and rent, £50, and petty cash and typiste, £23 10s. 7d., were passed for payment.

A report of the Annual Meeting of the Broken Hill board was submitted and received.

A request was submitted from members at Port Pirie for authority to form a branch, to include Whyalla and Iron Knob. The application was endorsed and recommended for approval by the Council.

Resolved that meetings of the Executive Committee be held on the last Monday in each month.

Applications for admission to the Institute were submitted and dealt with as follows:—As Associate Member—Mr. Howard R. Stafford (approved for ballot); as Junior Member—Mr. Leonard V. Ellis (approved for ballot).

The following were admitted as Students:—Messrs. A. H. D. Barratt, F. M. Jefferson, H. G. Pyke, C. E. Lambert, W. St. C. Osborne, G. R. Fisher, and M. A. Brooke.

It was reported that the following contributions towards preliminary expenses, Engineering Standardization, had been received:—Electrolytic Zinc Co. of Australasia, £25; Broken Hill Associated Smelters, £25.

Suggestions from Mr. H. W. Gartrell (Adelaide University) with regard to a scheme for forming Joint Student Sections in conjunction with the Institution of Engineers, Australia, was referred to a special committee for report to the Council.

Progress was reported in connection with the proposal that the Pan-Pacific Congress should include Broken Hill in its itinerary. Mr. E. C. Andrews, by letter, intimated that the visit, if arranged. would probably be made from Sydney, via Cobar. It was further reported that membership of the Pan-Pacific Congress was open to anyone interested on payment of 25s., and that institutes and societies were entitled to each nominate one delegate to the congress,

Mr. Wainwright invited the Council's assistance in the preparation of a list of persons engaged in scientific research work in mining and metallurgy, who should be invited to become "Associates" of the Australian National Research Council. Resolved that particulars be forwarded to all members of Council and to Branches, with the request that they suggest names of suitable men; such names to be forwarded to the Secretary Aust. I.M.M. for consideration by the Council.

Copy of amendments to regulations under the New South Wales *Mining Act* were received from Mr. G. H. Blakemore.

MAY 28TH, 1923—1 P.M.

The Secretary's report was presented, and accounts to the amount of £68, including postage, etc., £13, printing, £48, and auditing, £2 2s., were passed for payment.

Report of meeting of the Broken Hill Branch, held on 1st May, was submitted and received.

The following were admitted to the Institute:—As Associate Member—Mr. Howard Raymond Stafford; as Junior—Mr. Leonard Victor Ellis; as Student—Mr. George Roy Giles.

Applications for admission to the Institute as Associate Members from Messrs. Hugh Crawford and Samuel Hugh Sloan were approved for ballot.

A paper by F. L. Willan, entitled "The Geology and Ore-Deposits of Mount Stewart, Leadville, N.S.W.," was referred to the publication committee.

In view of the Prime Minister's recent favourable remarks in regard to the proposed Federal Geological Survey, it was agreed that the Australian National Research Council be recommended to now take steps to further the proposal.

Mr. H. W. Gepp (Vice-President) was nominated to represent the Institute on the Pan-Pacific Congress.

Resolved—"That the First Ordinary Meeting be held in December, commencing about the middle of the month; that a visit to the works of the Electrolytic Zinc Co. of Australia at Hobart, Tasmania, be a feature of the meeting."

Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power were appointed to represent the Institute on the Federal Engineering Council for the year commencing 1st July, 1923.

It was agreed that the Aus. I.M.M. co-operate with the Institution of Engineers, Australia, in approaching the Federal Government with the request that publications of technical bodies be carried at book-post rate.

Copy of minutes of second meeting of the Main Committee of the Commonwealth Engineering Standards Association were submitted and received.

———

JUNE 25TH, 1923—1 P.M.

The Hon. Treasurer's report was presented, and accounts to the amount of £141, including salaries and rent, £69, printing, £55, process blocks, £8, and postage, etc., £8, were passed for payment.

Resolved that Inscribed Stock maturing on 15th September be converted into the Commonwealth 5 % Loan maturing in 1948, and that interest on Inscribed Stock in future be paid to current account.

Resolved that stock be inscribed in the names of A. S. Kenyon and the Secretary instead of Mr. G. C. Klug and the Secretary.

As a result of the ballot by members of Council, the following were declared elected to the Institute as Associate Members :—Messrs. Hugh Crawford and Samuel Hugh Sloan.

The following nominations were approved for ballot :—As Members —Messrs. Arthur John Hemingway, John Clarence Hughes, and Cuthbert Edward Rees Jones; as Associate Members — Messrs. Gordon Colvin Lindesay Clark, Jack Esdale Parr Murrie, Joseph Frederick Smith, and Theodore d'Avelin Melhuish; as Juniors—Messrs. John Richard Allsop and Clarence Lewis Bowden Goddard. Messrs. Horace P. Buckley, Allen C. M'Donald, Kingsley H. Stacey, Reuben S. Russell, and William G. Sheil were admitted as Students.

Recommendation of the sub-committee in respect to joint students' sections was adopted.

A list of companies to be invited to subscribe to the cost of publications was submitted, and, after revision, adopted. The amount of subscription to be suggested was fixed at £10 per annum.

Resolved that the First Ordinary Meeting, 1923, at Hobart, Tasmania, commence on 17th December. The appointment of Mr. R. H. Berriman as local secretary to the meeting was endorsed, and instructions given *re* the appointment of a local committee.

Copies of letter received *re* proposed World Power Conference in 1924 were directed to be forwarded to members of Council for suggestions thereon.

Arrangements were made for the nomination of two representatives to the Electrical Standardization Sectional Committee, as invited by the Main Committee of the Commonwealth Engineering Standards Association.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

Since the last issue of Proceedings the following have been admitted to the Institute :—

Associate Members.

Names and Addresses.

Crawford, Hugh, School of Mines, Thames, N.Z.

Sloan, Samuel Hugh, c/o Bewick, Moreing and Co., 62 London Wall, London, E.C. 2.

Stafford, Howard Raymond, Wood's Point, Vic.

Junior.

Ellis, Leonard Victor, c/o Auckland Gas Co., Beaumont-street, Auckland, N.Z.

Students.

Barratt, Alfred Henry Douglas, Technical College, Broken Hill, N.S.W.

Buckley, Horace Patrick, 59 Sinclair-street, Woolstonecraft, N.S.W.

Brooke, Malcolm Archibald, St. Andrews-street, N. Walkerville, S.A.

Fisher, George Read, Fourth-avenue, Joslin, Adelaide, S.A.

Giles, George Roy, 4 Symonds-street, Auburn, Vic.

Jefferson, Francis Middleton, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.

Lambert, Charles Edward, "Mildred," Ninth-avenue, Campsie, N.S.W.

M'Donald, Allen Claude, 49 York-road, Pirie West, Port Pirie, S.A.

Osborne, Wilfred St. Clair, Ravensthorpe-avenue, Millswood Estate, Adelaide, S.A.

Pyke, Henry George, Cambridge-street, Penshurst, N.S.W.

Russell, Reuben Sussex, Manifold-street, Camperdown, Vic.

Sheil, William Glenister, 510 Rathdown-street, North Carlton, Vic.

Stacey, Kingsley Harvard, c/o Broken Hill Associated Smelters Pty. Ltd., Port Pirie, S.A.

FIRST ORDINARY MEETING, 1923, HOBART, TASMANIA.

(17th December to 22nd December.)

As previously announced, the First Ordinary Meeting, 1923, will be held at Hobart, Tasmania, commencing there on Monday, 17th December. Concessions in railway fares will be arranged for members and their wives. A visit to Hobart in December, under ordinary circumstances, is most enjoyable, and the programme of arrangements for this meeting, which includes an inspection of the works of the Electrolytic Zinc Co. of Australasia, Risdon-road Hobart, and the hydro-electric works at Great Lake, makes this visit a particularly attractive and interesting one. Owing to the heavy tourist traffic at that time of the year, and in order that hotel and boat accommodation may be secured, it will be necessary for members to notify their intention to be present before the end of October. All booking, steamer and rail, should be done through the Tasmanian Government Tourist Bureau, 59 William-street, Melbourne.

ELECTION OF PRESIDENT AND COUNCILLORS FOR 1924.

In accordance with Articles 38 and 40, the following officers retire at the end of 1923:—

President—Colin Fraser.

Councillors—Representing Australasia: Colin Fraser, James Hebbard, A. S. Kenyon, and W. E. Wainwright.
Representing New South Wales: C. C. Freeman and F. Danvers Power.
Representing Victoria: H. Herman.
Representing Queensland: W. H. Corbould.
Representing South Australia: H. Lipson Hancock.
Representing Western Australia: J. W. Sutherland.
Representing Tasmania: H. W. Gepp.
Representing New Zealand: D. B. Waters.

Nominations to fill the vacancies thus occurring are invited, and should reach the Secretary not later than 1st September, 1923. A retiring Councillor is eligible for re-election, and shall, unless he notifies his intention to withdraw, be deemed to be duly nominated for office.

PAN-PACIFIC CONGRESS.

Members are invited to join the Pan-Pacific Congress. The Melbourne session commences on 13th August and terminates on 22nd August. The Sydney session commences on 23rd August and terminates on 3rd September. The subscription is 25s. for full membership and 10s. for ladies and students. Application for membership should be made to the Hon. Secretary, Mr. R. H. Cambage, 5 Elizabeth-street, Sydney, or through the Secretary of the Aus. I.M.M. Sections likely to be of most interest to members are Section 8 (Geology) and Section 9 (Hygiene, including Mining Hygiene). A number of overseas geologists and hygienists will be present. Distinguished visitors, representing fifteen different countries, Dominions, and colonies, will attend. The Aust. I.M.M. is officially represented on the Congress by Mr. H. W. Gepp (Vice-President).

MOVEMENTS OF MEMBERS.

J. Atkinson is managing the Cherry Tin Mine, Ardlethan, N.S.W.

W. B. Chomley, for some time on the staff of the Zinc Corporation Ltd., Broken Hill, has joined the metallurgical staff of the British Broken Hill Co.

R. A. Clinton is with Mr. J. Atkinson on the Cherry Tin mine, Ardlethan, N.S.W.

G. A. Denny, who has been at Bulawayo, Rhodesia, for some time, has returned to England. His address is Salisbury House, London Wall, London, E.C. 2.

C. C. Freeman, superintendent of the Zinc Corporation, Broken Hill, left Australia for London in May. His address is c/o G. C. Klug, Bewick, Moreing and Co., 62 London Wall, London, E.C. 2.

G. C. Klug, manager in Australia for Bewick, Moreing and Co., sailed for London in May. His address now is c/o Bewick, Moreing and Co., 62 London Wall, London, E.C. 2. He was accorded a farewell dinner at Menzies' Hotel, Melbourne, on the eve of his departure by a large gathering of men representing metal interests.

C. O. G. Larcombe, of Kalgoorlie School of Mines and Acting Government Petrologist for W.A., has received the degree of Doctor of Science.

E. H. B. MACARTNEY, of Subiaco, W.A., visited Melbourne in June.

A. H. MERRIN, Chief Mining Inspector for Victoria, has been appointed Secretary for Mines.

HAROLD RABLING, late of Bonne Terre, U.S.A., has joined the staff of the Vacuum Oil Co., and is proceeding to Newcastle, N.S.W., to take up his duties there.

J. W. SALTER has been appointed an Inspector of Mines at Broken Hill, N.S.W.

H. R. STAFFORD, recently with the Morning Star mine, Wood's Point, Vic., is leaving for Burma on 20th July to join the staff of the Burma Corporation Ltd. at Nam Tu.

GEO. W. WRIGHT, who for many years has occupied the important position of chief mechanical engineer at Mount Lyell, Tasmania, has been appointed resident engineer of the National Portland Cement Co., Maria Island, Tasmania.

INDEX.

An index of Papers appearing in the Proceedings from No. 17, 1915, to No. 50, 1923, is published as a supplement to this number.

BRANCH REPORTS.

BROKEN HILL BRANCH.

A General Meeting of the Broken Hill Branch was held on Tuesday, 19th April, 1923, at 8 p.m. Mr. F. Voss Smith presided.

On behalf of Mr. James Hebbard, a paper describing the Coppus Vane blower used as a small ventilating unit was read by Mr. E. B. Dow. In the discussion which followed, Mr. Thos. Hanton gave a résumé of work carried out on the Zinc Corporation Mine, and Mr. R. T. Slee a description of ventilating units on the Broken Hill Proprietary mine. Mr. C. H. J. Clayton queried the effective water gauge of the unit, and drew attention to the fact that 1.6 in. of water gauge was required with about 200 ft. of 12-in. flexoid tubing. He stated that they were looking for a unit with a water gauge of 5 or 6 in.

The secretary outlined the replies received regarding the types of drill suitable for Broken Hill conditions, and asked for an opinion on the form in which they should be presented for publication. The matter was left in the secretary's hands.

A meeting of the Broken Hill Branch was held on 1st May, 1923, at 8 p.m. Mr. F. Voss Smith presided, and there was a good attendance.

The meeting was held for the purpose of bidding farewell to Mr. G. C. Klug and Mr. C. C. Freeman, who were shortly leaving for London.

Mr. Klug delivered an address on the subject of his recent visit to Europe. He outlined the methods and scope of activity in all the countries visited by him, and provided a most interesting evening.

On behalf of the branch, Mr. Voss Smith then bade Messrs. Klug and Freeman farewell.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

FROM 31ST MARCH, 1923, TO 30TH JUNE, 1923.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry: Journal	monthly	London
Chemical, Metallurgical and Mining Society of South Africa: Journal	monthly	Johannesburg
Franklin Institute: Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers: Journal	monthly	London
Queensland Department of Mines: Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines: Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines: Report of Executive	monthly	Bulawayo
Royal Society of Victoria: Proceedings, Vol. XXXV., Part II.		Melbourne
Australian National Research Council: Australian Science Abstracts, Vol. II., Nos. 1 and 2		Sydney
New South Wales Department of Mines: Bulletin Geological Survey, Nos. 2 and 3		Sydney
Royal Society of New South Wales: Journal and Proceedings, Vol. LV.		Sydney
South Australian Department of Mines: Mining Review, No. 37		Adelaide
The University of Adelaide: Calendar, 1923		Adelaide

School of Mines, Adelaide :	
Prospectus, 1923 ; Annual Report, 1922	Adelaide
Government Geologist of West Australia :	
Annual Progress Report of Geological Survey, 1922	Perth
The University of Tasmania :	
Calendar, 1923	Hobart
New Zealand Board of Science and Art :	
Journal, Vol. V., No. 6 ; Vol. VI., No. 1	Wellington
School of Mines, Waihi, New Zealand :	
Syllabus, 1922	Waihi
Geological Survey of New Zealand :	
Bulletins, Nos. 24 and 25	Wellington
Institution of Mining and Metallurgy :	
Bulletins, Nos. 223 and 224	London
Institution of Mining Engineers :	
Transactions, Vol. LXIV., Part 6 ; LXV., Parts 1 and 2	London
Geological Society :	
Quarterly Journal, No. 313	London
Geologists' Association :	
Proceedings, Vol. XXXIV., Parts 1 and 2 Index to Vols. XXI.-XXX.	London
Imperial Mineral Resources Bureau:	
Reports on Strontium, Silver and Molybdenum	London
Annales des Mines :	
1923, Parts 3-5	Paris
Department of Mines in Mysore :	
Report of Chief Inspector, 1920-21, 1921-22	Bangalore
Department of Mines, Canada (Mines Branch) :	
Tatanium by A. H. A. Robinson	Ottawa
Department of Mines, Toronto, Ontario :	
Thirteenth Annual Report, 1921, Parts I.-VII.	Toronto
American Mining Congress :	
Third Standardisation Bulletin, 1923	New York
Geological Survey of Alabama :	
County Report, No. 1	Alabama

California State Mining Bureau :
  Bulletin, No. 91 - - - - Sacramento
  "Mining in California" (monthly), "Summary of Operations, California Oil Wells" (monthly) - San Francisco

Secretaria de Industria Y Comercio :
  Bulletins, Vol. XIV., Nos. 2 and 3 - - Mexico

Instituto Geologico de Mexico :
  Bulletin, No. 38 - - - - Mexico

Department of the Interior—Bureau of Mines :
  Technical Papers, Nos. 290, 301 and 309 -
  Bulletins, Nos. 186, 187, 199 and 202 - Washington

RECENT ARTICLES ON MINING MATTERS.

(31st March, 1923 to 30th June, 1923.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.
(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.
(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.
(4) *The Mining Journal*, London, E.C., wk., 6d.
(5) *Engineering and Mining Journal Press.*
(6) *Annales des Mines*, Paris, France, mth.
(7) *Publications*, Department of Mines, Melbourne, Victoria.
(8) *Publications*, Department of Mines, Sydney, New South Wales.
(9) *Publications*, Department of Mines, Adelaide, South Australia.
(10) *Publications*, Department of Mines, Brisbane, Queensland.
(11) *Publications*, Department of Mines, Perth, Western Australia.
(12) *Publications*, Department of Mines, Hobart, Tasmania.
(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.
(14) *Publications*, Bureau of Mines, Toronto, Ontario.
(15) *Publications*, Geological Survey of India, Calcutta.
(16) *Publications*, Geological Survey, U.S.A., Washington.
(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.
(18) *Publications*, California State Mining Bureau, Sacramento, Cal.
(19) *Reports* Aust. Assoc. Adv. Science, Sydney, New South Wales.
(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.
(21) *Quarterly Journal*, Geological Society, London.
(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.
(23) *Transactions*, Inst. Min. Eng., London.
(24) *Journal*, Canadian Mining Inst., Ottawa, Ontario.
(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.
(26) *Transactions*, Am. Inst. of Min. & Met. Eng. New York City.
(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.
(28) *Journal*, Franklin Inst., Philadelphia, Pa.
(29) *Chemical Engineering and Mining Review.*
(30) *Transactions*, Am. Soc. C.E., New York City.
(31) *Bulletins*, Société des Ingénieurs Civils, Paris.
(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.
(33) *Publications*, Iron and Steel Institute, London.
(34) *Proceedings*, Inst. of Mech. Eng., London
(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.
(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.
(37) *Transactions*, Mining and Geological Institute of India, Calcutta.
(38) *Publications*, Department of Mines, Wellington, N.Z.
(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.
(40) *Publications*, Bureau of Mines, Dept. of the Interior, Washington, U.S.A.

LIST OF ARTICLES.

MECHANICAL.

Inclusions in Aluminium-Alloy Sand Castings. R. J. Anderson. (40) Technical Paper No. 290.

Rock-Drill Steel. R. J. Day. (5) April 14, 1923.

Useful Fan Calculations. W. S. Weekes. (5) April 14, 1923.

Heat-Treating Drill Steel. F. B. Foley. (5) May 12, 1923.

Abuse of Wire Ropes. H. M. Hall. (5) May 19, 1923.

Standardizing Drill Steel at the Copper Queen Mines. A. Notman. (5) June 2, 1923.

Rotation Mechanisms for Rock Drills. R. J. Day. (5) June 9, 1923.

METALLURGICAL.

Proposed Method of Reducing Mineral Waste in the Wisconsin Zinc District. W. H. Coghill and C. O. Anderson. (40) Technical Paper 301.

Experimental Production of Alloy Steels. H. W. Gillett and E. L. Mack. (40) Bull. No. 199.

Treatment of the Tungsten Ores of Boulder County, Colo. J. P. Bonardi and J. C. Williams. (40) Bull. No. 187.

Electric Brass Furnace Practice. H. W. Gillett and E. L. Mack. (40) Bull. No. 202.

Investigation of Zirconium, with Especial Reference to the Metal and Oxide. J. W. Marden and M. N. Rich. (40) Bull. No. 186.

Pyritic Smelting in a Reverberatory, Defended by its Sponsor. E. C. King. (5) April 21, 1923.

Ore Dressing, Fluxes, and Fuels at Namtu. R. G. Hall. (5) April 21, 1923.

The Lead Blast-Furnace Department of the Great Falls Zinc Plant. R. B. Caples. (5) April 21, 1923.

Smelting Practice at Namtu. R. G. Hall. (5) April 28, 1923.

Determination of Vanadium in Ores. A. H. Low. (5) April 28, 1923.

The Laws of Crushing. H. Hardinge. (5) May 5, 1923.

Regeneration of Cyanide Solutions. H. B. Wright. (5) May 5, 1923.

Cyaniding Gold Ore at the Ashanti Mine. (5) May 12, 1923.

The Reploglc Magnetic Tailing Re-treatment Plant. W. R. Canton. (5) May 12, 1923.

Use of Peroxides in the Cyanide Process. W. J. Starwood. (5) May 19, 1923.

The Wright-Hargreaves Cyanide Mill. J. E. Grant. (5) May 19, 1923.

Amalgamation and All-Cyanidation. H. Hanson. (5) May 19, 1923.

Magnetic Separation of Complex Lead-Zinc Ores. E. Hedburg. (5) May 26 1923.

New Magnetic Concentrating Mill at the Richard Iron Mine, New Jersey. H. M. Roche. (5) Serial commenced May 26, 1923.

Safety Locking Device for Filter Press. W. J. Tait. (5) May 26, 1923.

Apparatus for Aeration of Solutions. R. V. Thurston. (5) May 26, 1923.

The Magnetic Separation of Pyrite Blende. W. R. Wade. (5) June 16, 1923.

The Equilibrium Diagram of the System Cu_2-FeS. C. B. Carpenter and C. R Hayward. (5) June 16, 1923.

The Dielectric Constant as a Factor in Ore Concentration. B. W. Holman. (32) May, 1923.

The Titration of Very Small Amounts of Gold. W. B. Pollard. (22) April, 1923.

Notes on the Selection of a Copper Smelting Plant. W. A. Heywood. (5) April, 1923.

The Solubility of Gold Amalgam in Cyanide Solutions. H. A. White. (25) March, 1923.

Observations on the Tin-Antimony Alloys. R. W. Irwin. (25) March, 1923.

The Recovery of Gold Absorbed in Reduction Works. A. King. (25) May, 1923.

MINING.

Recent Progress in the Thawing of Frozen Gravel in Placer Mining. C. Janin. (40) Technical Paper No. 309.

Mining Phosphate Rock in Florida. H. M. Payne. (5) April 21, 1923.

A Graphical System of Picturing Mining Methods. C. M. Haight. (5) May 5, 1923.

Improved Safety Crosshead for Bucket Shafts. H. F. Lunt. (5) May 12, 1923.

Shaft-Signalling System at Ironton Mine, Bessemer, Mich. W. H. Carrick. (5) May 12, 1923.

Effect of Increasing Depth of Hole in Stoping. H. W. Seamon. (5) June 2, 1923.

Notes on Fan Ventilation in Deep Mines. J. A. Woodburn. (25) May, 1923.

GEOLOGY.

Geology and Mineral Resources of Clay County, with Special Reference to the Graphite Industry. W. F. Prouty. (17) County Report No. 1.

Geology of the Homestake Ore-Bodies and the Lead Area of South Dakota. J. O. Hostet and L. B. Wright. (5) Serial commenced May 5, 1923.

MISCELLANEOUS.

Transportation, Power, and Mining at Bawdwin. R. G. Hall. (5) April 14, 1923.

The Marketing of Copper. E. H. Robie. (5) April 21, 1923.

The Marketing of Bauxite. J. M. Hill. (5) May 5, 1923.

The Marketing of Selenium and Tellurium. F. F. Colcord. (5) May 19, 1923.

The Marketing of Lead. I. H. Cornell. (5) June 2, 1923.

The Marketing of Phosphate Rock. R. W. Stone. (5) June 16, 1923.

Sorting, Hauling, and Sampling Carnotite Ore. H. S. Worcester. (5) June 16, 1923.

A Proposed Simple Alternative Method of Testing Mine Dusts. J. W. Whitaker. (23) Vol. LXIV., Part 5.

The Possibility of Spontaneous Combustion Being Initiated by the Heat Produced in Crushing. H. Briggs. (23) Vol. LXIV., Part 5.

A Contribution to the Theory of Subsidences. H. Louis. (23) Vol. LXIV., Part 5.

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Notes on the Testing of Mine Dusts. W. M'D. Mackey. (23) Vol. LXV., Part 1.

A Rapid Volumetric Method for the Determination of Carbonic Acid in Mine Dusts Containing Carbonates. W. M'D. Mackey and W. Ward. Vol. LXV., Part 1.

Report on the Coal-Dust Experiments at the Mines Department Experimental Station, Eskmeals, Cumberland. J. R. Felton, L. Holland, and H. H. Ridsdale. (23) Vol. LXV. Part 1.

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OF THE
AUSTRALASIAN INSTITUTE OF MINING & METALLURGY
(INCORPORATED).

¶ Life member. F Foundation member. § Author of paper.

MEMBERS.

Names and Addresses.	Date of Election
Adam, John, Sons of Gwalia Mine, Gwalia, W.A.	1910
Adams, Ernest F., Box 5, P.O., Thames, N.Z.	1900
Adams, James Henry, Thames, N.Z.	1911
Agnew, John A., 1 London Wall Buildings, London, E.C. 2	1909
§Allen, Francis Bowen, 38 Colin-street, West Perth, W.A.	1900
Anderson, J. R. V., School of Mines, Bendigo, Vic.	1910
Andrews, E. C., Mines Department, Sydney, N.S.W.	1921
§Aplin, Wm., Mt. Morgan Technical College, Mt. Morgan, Q.	1905
§Atkinson, A. A., "Ingleholme," Northcote-road, Lindfield, N.S.W.	1912
§Avery, David, Collins House, 360 Collins-street, Melbourne, Vic.	1920
Avery, John, 4 Kelvin-grove, Armadale, Vic.	1918
Baker, William H., School of Mines, Thames, N.Z.	1911
Banks, Charles A., 36 Kensington-place, Nicola-street, Vancouver, B.C., Canada	1911
§Banks, Edwin G., Box 28, P.O., Waihi, N.Z.	1910
Baragwanath, Wm., Geological Survey Office, Mines Depart., Melbourne, Vic.	1915
Barker, Geo., Mt. Rosebery Mines, Rosebery, Tas.	1915
Barr, James (jun.), c/o J. A. Russell & Co., Kuala Lumpur, Fed. Malay States	1913

Names and Addresses.	Date of Election.
Barron, William Ernest, c/o National Bank of South Africa, Simmonds-street, Johannesburg, South Africa	1921
Barton, B. V., 323 Glenferrie-road, Malvern, Vic.	1910
Basedow, Erwin, Pirie-street, Kent Town, S.A.	1906
Bassett, John, Hall-street, Hokitika, N.Z.	1911
Beamish, W. A., c/o Briseis Company, Derby, Tas.	1904
§Beaumont, Edwin K., "Kenilworth," 24 Lewis-street, Lamington Heights, Kalgoorlie, W.A.	1899
Beech, Harold, Sons of Gwalia Mine, Gwalia, W.A.	1919
§Bellinger, H. C., 25 Broadway, New York, U.S.A.	1910
Bennett, V. C., c/o Electrolytic Zinc Co. of Australasia Ltd., Collins House, 360 Collins-street, Melbourne, Vic.	1914
Bennetts, A. T., Mines Office, Bentong, Pahang, Fed. Malay States	1923
Bennie, Boyd, First Avenue. Whargarei, N.Z.	1911
Blackett, Charles E., Golden Horseshoe Estates Co. Ltd., Boulder, W.A.	1909
¶§Blakemore, G. H., Commercial Bank Chambers, 273 George-street, Sydney, N.S.W.	F.
§Bohm, Horace Sanford, "The Range." Rockhampton, Q.	1898
§Bowater, W. H., Box 9, Kuridala, via Cloncurry, Q.	1910
Boyd, Adam A., c/o Mount Morgan G.M. Co., Ltd., Mount Morgan, Q.	1906
Bradford, Leslie, Broken Hill Pty. Ltd., Iron and Steel Works, Newcastle, N.S.W.	1910
Bray, G. E., Charters Towers, Q.	1908
Brett, Henry Tyndall, Albany Chambers, Bulawayo, Rhodesia, South Africa	1912
§Bridge, John M., 3 Testar-grove, Caulfield, Vic.	1920
§Brinsden, Frederick G., South Kalgurli Ltd., Fimiston, W.A.	1909
¶§Broome, George H., State Coal Mine, Wonthaggi, Vic.	1900
§Brown, George F. Campbell, Main Roads Board, State Insurance Building, Brisbane, Q.	1910
Brown, H. Wheeler, 434 Collins Street, Melbourne, Vic.	1913
Brown, Lionel Venn, "Ocean House," Moore-street, Sydney, N.S.W.	1920
§Browne, D. H., "Bonnie Doon," Wonthaggi, Vic.	F.
Buckley, Henry, Zinc Corporation Ltd., Broken Hill, N.S.W.	1901

Names and Addresses.	Date of Election.
Butement, Thomas, School of Mines, Kalgoorlie, W.A.	F.
Cameron, Henry Lovett, 5 Hamilton-street, Sydney, N.S.W.	1922
§Campbell, A. G., Block 10, Broken Hill, N.S.W.	1922
Campbell, John Kenneth, Stannary Hills, via Cairns, Q.	1921
§Campbell, William Horace, c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1920
Campion, G. W., Tarquah, Gold Coast, West Africa	1901
Carroll, John J., Kuaotunu, Auckland, N.Z.	1911
Chambers, Preston, 22 Shortland-street, Auckland, N.Z.	1912
§Chapman, Professor R. W., The University, Adelaide, S.A.	1901
Chapple, Albert J. M., "Exeter," Barker-road, Strathfield, N.S.W.	1906
Chapple, William Henry Charles, 49 Smith-street, N.S.W.	1921
Chomley, W. B., British Australian Broken Hill Ltd., Broken Hill, N.S.W.	1912
Church, Kenneth C., Broken Hill Pty. Ltd., Newcastle, N.S.W.	1913
Clark, Lindesay C., Derby, Tas.	1905
Clarke, Huntley J., Queenstown, Tas.	1901
Clayton, C. H. J., c/o Zinc Corporation Ltd., Broken Hill, N.S.W.	1913
§Clayton, H. Edward, New Guinea Copper Mines Ltd., Port Moresby, Papua	1920
§Cleland, W. Lauder, Broken Hill Pty. Ltd., Iron and Steel Works, Newcastle, N.S.W.	1907
Clinton, Robert A., c/o Cherry Tin Mine, Ardlethan, N.S.W.	1921
Colley, Walter, c/o Messrs. Fearnley and Co., Cairns, Q.	1914
Conder, Hartwell, Strahan, Tas.	1915
Connor, J. D., Wonthaggi Technical College, Wonthaggi, Vic.	1910
Conway, James, State Coal Mine, Bowen, Q.	1921
§Corbould, W. H., Stanway House, 77 King-street, Sydney, N.S.W.	1908
§Courtney, Charles Frederick, Finsbury House, Blomfield-street, London, E.C, 2.	1901
§Cowles, Richard K., New Sylvia G.M. Co., Box 55, P.O., Thames, N.Z.	1911
Cranz, William Nelson, Broken Hill Proprietary Ltd., Broken Hill, N.S.W.	1921
Crozier, Robert H., P.O. Box 291, Rangoon, Burma	1913

Names and Addresses.	Date of Election
Crutchett, Edgar A., Great Boulder G.M. Co., Boulder, W. A.	1909
Currie, David, 276 Salisbury House, London Wall, London, E.C. 2.	1911
Daggar, H. James, c/o Stock Exchange Club, Collins-street, Melbourne, Vic.	1901
§Davies, H. Warlow, c/o Electrolytic Zinc Co. of Australasia Ltd., Risdon, Tas.	1910
Dawson, Edward H., 18 Exchange, Broken Hill, N.S.W.	1910
§Delprat, G.D., "Keynsham," Mandeville Crescent, Toorak, Vic.	1902
§Denny, G. A., Salisbury House, London Wall, E.C. 2	F.
Dey, George, Balmoral-road, Port Pirie, S.A.	1920
§Dibdin, T. R., Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1910
Dickson, Gordon F., London & Rhodesian Mining & Land Co., 5 Moorgate-street, London	1909
Dixon, Clement, Cyprus Asbestos Co. Ltd., Amandos, via Limassol, Cyprus	1900
¶Dobbie, H. Campbell, Mergui, Burma, India	1911
Dods, Richard, c/o Mount Lyell Mining and Railway Co., Queenstown, Tas.	1910
Donaldson, James, Box 272, P.O., Cairns, North Q.	1914
§Donaldson, Robert J., Sulphide Corporation Ltd., Boolaroo, N.S.W.	1905
Doolette, Dorham L., Nat. Mut. Buildings, St. George's-terrace, Perth, W.A.	1909
Drage, James, Ivanhoe Gold Corporation, Boulder, W.A.	1909
§Driffield, E. Carus, Queenstown, Tas.	F.
Dubois, Alex. J., "Walhalla," Fernsplatt, Perranwell, S.O., Cornwall, England	1908
Dunn, A. C., Central Mine, Broken Hill, N.S.W.	1908
Dyason, Edward C., Equitable Building, Melbourne, Vic.	1911
§Dyson, Thomas Ingleby, Collins Court, 374 Little Collins-street, Melbourne, Vic.	F.
Eastaugh, Frederick A., University Club, Sydney, N.S.W.	1910
Eaton, J. Murdoch, Royalist Mine, Que Que, Rhodesia, South Africa	1905
§Edmands, H. R., Riverina South Co., via Menzies, W.A.	1901

Names and Addresses.	Date of Election.
Edquist, Victor T., Big Hill Mine, Cue, W.A.	1909
Emery, Cyril J., British Mine, Broken Hill, N.S.W.	1919
Espie, Frank F., Burma Mines Ltd., Bawdwin, Northern Shan States, Burma, India	1918
Evans, H. A., Sulphide Corporation Ltd., Boolaroo, N.S.W.	1911
Ewart, John, North Broken Hill Ltd., Broken Hill, N.S.W.	1910
§Fairweather, Andrew, Broken Hill South Ltd., Broken Hill, N.S.W.	1912
Farrier, William Charles Valentine, P.O. Box 28, Broken Hill, N.S.W.	1913
Faul, Charles, "Moola," St. Leonard's-avenue, St. Kilda, Vic.	1910
Finch, Edward W., 12 Loftus-street, Sydney, N.S.W.	1922
Fisher, D. H., Mount Morgan, Q.	1907
Foster, Donald F., The Cliffe, Belmont, Sutton-Surrey, England	1913
Fraser, Alex., Great Boulder Pty. Ltd., Boulder, W.A	1909
Fraser, Colin, c/o Broken Hill Associated Smelters Ltd., Collins House, 360 Collins-street, Melbourne, Vic.	1911
Fraser, Donald Lovat, Rockhampton Club, Rockhampton, Q.	1910
Fray, Percival Richard, St. Arnaud, Vic.	1910
§Freeman, Charles C., c/o Bewick, Moreing & Co., 62 London Wall, London, E.C. 2	1913
Gall, Wm. C., North Broken Hill Ltd., Broken Hill, N.S.W.	1910
Garde, Henry T., Irrigation Trust, Barmera, S.A.	1913
Gardner, W. E., Central Mine, Broken Hill, N.S.W.	1902
Garland, Charles L., 11 Moore-street, Sydney, N.S.W.	1898
§Gartrell, H. W., The University, Adelaide, S.A.	1910
Gepp, Herbert W., Electrolytic Zinc Co. of Aust. Ltd., Collins House, 360 Collins-street, Melbourne, Vic.	1910
Gibson, Angus A., Mount Morgan, Q.	1910
§Gilmour, Jas. L., Waihi, N.Z.	1901
Godden, Fred. W. R., Hampton Properties Ltd., Block 50, Hampton Plains, W.A.	1911
Gosman, William A., Queenstown, Tas.	1913
Gray, George J., Kerry Cottage, Arkaringa Crescent, Black Rock, Vic.	1912
Greig, E. H. c/o Burma Mines Ltd., Nam Tu, Northern Shan States, Upper Burma	1916

Names and Addresses.	Date of Election.
Grierson, J. J., c/o South Australian Cement Co., Brighton, Adelaide, S.A.	1911
Grigg, Joseph, The Malayan Prospecting Syndicate Ltd., Kuantan, via Singapore, Fed. Malay States ..	1920
§Gunn, Francis, Calton-road, Gawler, S.A.	1910
Hack, Clem. A., 360 Collins-street, Melbourne, Vic. ..	1905
Haigh, Victor A., Broken Hill Proprietary Block 10 Ltd., Broken Hill, N.S.W.	1920
Hales, Robert G., Magnet, Tas.	1914
Hall, Ernest K., Christmas Gift G.M. Co., N.L., Cullinga Mine, P.O. via Wallendbeen	1920
§Hall, John F., 14 New Zealand Chambers, Queen-street, Brisbane, Q.	1918
Hamilton, Henry J. E., British Australian Broken Hill Ltd., North Broken Hill, N.S.W.	1910
§Hamilton, Richard, Great Boulder Pty. G.M. Co., Boulder, W.A.	1906
§Hancock, H. Lipson, Gawler Park, Angaston, S.A. ..	F.
Hancock, H. Ley, Nungarin, via Merredin, W.A. ..	1901
Hansen, L., North Ajax Mine, Daylesford, Vic.	1908
§Hanton, Thos. G., c/o Zinc Corporation Ltd., Broken Hill, N.S.W.	1919
Hargraves, Ernest P., c/o David & Toft, Singapore, Fed. Malay States	1915
§Hart, G. Stephen, c/o Commonwealth Cement Co., Portland, N.S.W.	1910
Harrison, Hector J., 656 Chapple-street, Broken Hill, N.S.W.	1921
Hasson, Robt., Hagelthorne-street, Wonthaggi, Vic. ..	1915
Hay, Archibald L., 25 Douglas-street, Toorak, Vic. ..	1909
§Hebbard, James, Central Mine, Broken Hill, N.S.W. ..	1902
§Hemingway, Arthur J., Broken Hill Associated Smelters, Port Pirie, S.A.	1923
Henderson, Edwin T., Broken Hill Pty. Ltd., Broken Hill, N.S.W.	1910
§Herman, Hyman, Electricity Commission, William-street, Melbourne, Vic.	1897
Hetherington, J. W., M. L. A. Chambers, Queen-street, Brisbane, Q.	1911

Names and Addresses.	Date of Election.
Hills, Loftus, Geological Survey Office, Launceston, Tas.	1921
§Hitchcock, W. E., Bismuth, Tin and Wolfram Mine, Moina, near Wilmot, Tas.	1904
Hockey, Francis R., c/o Broken Hill Pty. Ltd., Newcastle, N.S.W.	1922
Hofman, Prof. H. O., Massachusetts Institute of Technology, Boston, Mass., U.S.A.	1914
Honman, C. Sidney, Heidelberg, Vic.	1920
Horsburgh, James, c/o Mount Morgan G.M. Co. Ltd., Mount Morgan, Q.	1908
§Horwood, E. J., c/o Broken Hill Pty. Ltd., Equitable Building, Melbourne, Vic.	1902
Hosking, Arthur F., Champion Reefs, Mysore State, India	1920
Howe, A. Maxwell, Department of Mines (State Batteries Branch), Perth, W.A.	1901
§Hughes, John C., 8 Thomas-street, Port Pirie, S.A.	1923
Hunter, Peter, Wonthaggi, Vic.	1912
§Hunter, Stanley B., Mines Department, Melbourne, Vic.	1894
Hylton, C. G., Zinc Corporation Ltd., Broken Hill, N.S.W.	1906
Ikeda, Kenzo, Kosaka Copper Mines, Akita-Ken, Japan	1918
Imanaga, I., Müke, Japan	1917
§Ireland, Mark, South Mount Cameron, Tas.	1902
§Jakins, G. Fred., Mount Lyell Mining Railway Co., Gormanston, Tas.	1910
James, Eric L., P.O., Lake Boga, Vic.	1910
James, John C., "Boolgatta," Sandridge-street, Bondi, Sydney, N.S.W.	1913
§Jarman, Arthur, 1001 Spruce-street, Berkeley, Cal., U.S.A.	1910
§Jensen, E. Broughton, South Australian Railways, Resident Engineer's Office, Bordertown, S.A.	1909
§Jensen, William S., Mount Morgan Mining Co. Ltd., Mount Morgan, Q.	1908
§Johns, Frederick C., c/o White & Hosier, Broken Hill, N.S.W.	1911
Johns, Morgan J., Box 2, Mount Morgan, Q.	1909
§Johnston, W. H., Waihi G.M. Co. Ltd., Waihi, N.Z.	1904
Joice, John, Raub Australian G.M. Co. Ltd., Raub, Pahang, Fed. Malay States	1918

Names and Addresses.	Date of Election.
Jones, Henry, Inspector of Mines, Kadina, S.A.	1910
Jones, C. E. Rees, Broken Hill Associated Smelters, Port Pirie, S.A.	1923
Judge, James, Ravenswood, North Q.	1911
Julius, Archibald C., Ottery Mine, Tent Hill, Deepwater, N.S.W.	1919
§Kenyon, A. S., State Rivers and Water Supply Commission, Treasury Buildings, Melbourne, Vic.	1896
Key, John F., British Australian Broken Hill Ltd., Broken Hill, N.S.W.	1912
King, A. J., Taiping, Perak, Fed. Malay States	1911
Kingston, Wm., Bairnsdale, Vic.	1914
§Kitson, Albert E., 29 Albert-place, South Kensington, London, S.W. 7	1904
Klug, Geo. C., c/o Bewick, Moreing and Co., 62 London Wall, London, E.C. 2.	1909
§Larcombe, C. O. G., School of Mines, Kalgoorlie, W.A.	1908
Laun, Ernest J., Charters Towers, Q.	1909
§Lavers, Herbert, Block 14 Mine, Broken Hill, N.S.W.	1910
Learmont, Tom, Broadway, Reefton, N.Z.	1911
Leggo, A. Victor, 497-503 Collins-street, Melbourne, Vic.	1908
Leggo, H. Maddron, c/o A. Fraser, Esq., 210 Creswick-road, Ballarat, Vic.	1901
Lester, Thomas, Mount Morgan Extension Co. Ltd., Mount Morgan, Q.	1911
Lewis, Essington, c/o Broken Hill Pty. Ltd., 320 Collins-street, Melbourne, Vic.	1918
§Lewis, James B., Jessie-street, East Malvern, Vic.	1906
Lilburne, Albert S., Raub Australian G.M. Co., Raub, Pahang, Fed. Malay States	1911
Lindsay, John A., Broken Hill, N.S.W.	1906
Loring, W. J., 614 Crocker Building, San Francisco, California, U.S.A.	1909
§Low, V. F. Stanley, c/o National Bank of Australasia, 7 Lothbury, London, E.C.	1902
Lowry, Alfred, c/o North Broken Hill Ltd., Broken Hill, N.S.W.	1923
Lund, Jens, 64 Clyde-road, Addiscombe, Surrey, England	1902

Names and Addresses	Date of Election.
Macartney, E. H. B., Bagot-road, Subiaco, W.A.	1901
Mackay, Alexander D., Wombat Mines, Waratah, Tas.	1920
Mackenzie, D. C., Riverina Colleries, Oaklands, N.S.W. ..	1910
Macleod, Peter J., The University, Hobart, Tas. ..	1901
Macnicol, A. N., 31 Queen-street, Melbourne, Vic. ..	1906
Marryat, Cyril B., Metallurgy Department, National Physical Laboratory, Teddington, Middlesex, England ..	1910
§Mars, F. J., Broken Hill South Ltd, Broken Hill, N.S.W. ..	1902
Marshall, Reginald R., c/o Sulphide Corporation Ltd., 34 Queen-street, Melbourne, Vic.	1910
Martin, Edward, Shire Hall, Bright, Vic.	1902
§Mathers, J. V. W., Light of Asia Mine, Cue, W.A. ..	1914
§Matters, Clem. W., "Boongala," Chapel-street, Glenelg, S.A.	1910
§Mawdsley, W. H., 24 Barnard-street, North Adelaide, S.A.	1898
§M'Bride, Wm. J., Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1920
M'Conachie, Wm., Waihi Grand Junction Co., Waihi, N.Z. ..	1910
M'Cormick, W. J., Thames, N.Z.	1912
M'Geachie, Duncan, "Craig Royston," Toronto, N.S.W. ..	1912
M'Geachie, John, 28 O'Connell-st,, Sydney, N.S.W. ..	1908
§M'Keown, Maurice R., Valla Gold Mines, Urunga, N.S.W.	1912
§M'Leish, John, Wonthaggi, Vic.	1912
M'Padden, J. F., School of Mines, Westport, N.Z. ..	1911
Meredith, David, Electrolytic Zinc Co. of Australasia Ltd., Hobart, Tas.	1910
Merrill, Chas. W., Room 502, 121 Second-street, San Francisco, Cal., U.S.A.	1908
Merrin, Arthur H., Mines Department, Treasury Buildings, Melbourne, Vic.	1903
Messenger, William T., c/o Zinc Corporation Ltd., Broken Hill, N.S.W.	1913
Miller, Joseph, Story's Creek Tin Mining Syndicate, Avoca, Tas.	1920
Mitchell, John P., Atlas Buildings, 8 Spring-street, Sydney, N.S.W.	1909
§Mitchell, S. R., Burnley-street, Burnley, Vic.	1916
Moen, Oluf, Waitahuna, Otago, N.Z.	1911

Names and Addresses.	Date of Election
Moline, Arthur H. P., c/o Edward Dyson & Co., Equitable Building, Melbourne, Vic.	1909
Molineaux, Herbert S., The Reefs, Central Otago, N.Z.	1911
Moncrieff, P. W., Zinc Corporation Ltd., Broken Hill, N.S.W.	1919
§Montgomery, A., Department of Mines, Perth, W.A.	F.
More, George A., c/o Sime, Darby and Co. Ltd., Singapore	1920
§Morgan, Percy G., Geological Survey Office, 38 The Terrace, Wellington, N.Z.	1900
Morison, David N., Newcastle, N.S.W.	1920
Morse, Phillip S., 157 Walnut-street, Brooklyn, Mass., U.S.A.	1905
§Moss, Frank A., 7 Dalgety Buildings, William-street, Perth, W.A.	F.
Moulden, J. C., Central Zinc Co., Seaton-Carew, near Middlesborough, England	1906
§Moule, John W., Herberton, North Q.	1908
Moxon, Claude H. Hercules Gold and Silver Mining Co., Williamsford, Tas.	1915
Mulligan, Edric N., Austral Malay Tin Ltd., Taiping, Fed. Malay States	1920
Mulligan, W. J., Grafton Copper Mines, Cangai, N.S.W.	1910
Murdoch, Fredk. M., Waterloo Chemical Works Ltd., Box 444, Hobart, Tas.	1915
Murphy, Frank J., Great Cobar Ltd., Cobar, N.S.W.	1912
Murray, Geo. D., Ivanhoe Gold Corporation Ltd., Boulder, W.A.	1909
§Murray, Russell M., Mount Lyell M. & R. Co., Queenstown, Tas.	1911
Murray, Virginius H. R., Golden Horseshoe Estates Co., P.O. Box 78, Boulder, W.A.	1909
§Newman, J. Malcolm, Pratten Buildings, 26 Jamieson-street, Sydney, N.S.W.	1910
Nicol, W. J., United Gleeson's G.M. Co., Ten Mile, via Jamieson, Vic.	1910
Nicholson, L'Estrange, Mount Elliott Mine, Selwyn, Q.	1913
Noble, Jas. R., Muir's Gold Reefs Ltd., Te Puke, Bay of Plenty, N.Z.	1910
O'Brien, Kirk, 360 Collins-street, Melbourne, Vic.	1911

Names and Addresses.	Date of Election.
Osborne, Percy R., Bautama, Bootless Inlet, via Port Moresby, Papua	1921
Owen, Tom Mackellar, c/o E. W. Culver, 116. 118 Clarence-street, Sydney, N.S.W.	1914
§Palmer, Thos. H., Collins House, 360 Collins-street, Melbourne, Vic.	1910
Parsonage, William, State Coal Mine, Runanga, Greymouth, N.Z.	1920
§Patterson, B. G., Mount Morgan, Q.	1907
Paul, Matthew, Inspector of Mines, Waihi, N.Z.	1910
§Paull, W. J., "Codrington," Bowenville, Darling Downs, Q.	F.
Payne, Professor Henry, The University, Melbourne, Vic.	1911
§Pearce, W. C. Walworth, Wondecla, Herberton, Nth. Q.	1906
Pearson, J. L., Wallaroo Mines, S.A.	1913
Peberdy, James R., Rawang Tin Ltd., Selangor, Fed. Malay States	1910
Penman, Arthur Percy, Intercolonial House, Castlereagh-street, Sydney, N.S.W.	1911
Pomroy, H., Wallaroo Mines, S.A.	1913
§Poole, Wm., Culwulla Chambers, Castlereagh-street, Sydney, N.S.W.	1905
§Pound, J. R., 601a Dana-street, Ballarat, Vic.	1920
§Power, F. Danvers, 68½ Pitt-street, Sydney, N.S.W.	F.
Rae, Cecil, Ipoh, Perak, Fed. Malay States	1908
§Read, Thomas A., c/o Broken Hill South Ltd., Broken Hill, N.S.W.	1918
Reid, A. M'Intosh, Geological Survey Office, Launceston, Tas.	1919
Reid, Major D. E., Gympie, Q.	1910
Reid, Fredk. W., S.A. School of Mines, Adelaide, S.A.	1923
Rhodes, Charles, Ronaki, Remuera, Auckland, N.Z.	1911
§Rhodes, Fred. N., c/o Wilson's (N.Z.) Portland Cement Ltd., Portland, N.Z.	1910
Rickleman, H., c/o Electrolytic Refining Co., Port Kembla, N.S.W.	1912
Ridge, H. M., 2 Gt. Winchester-street, London, E.C.	1901
§Rigby, E. J., 482 Collins-street, Melbourne, Vic.	F.
Rigg, Gilbert, c/o Broken Hill Associated Smelters Ltd., Collins House, 360 Collins-street, Melbourne, Vic.	1917

Names and Addresses.	Date of Election.
Roberts, G. M., Associated Northern Blocks (W.A.) Ltd., Kalgoorlie, W.A.	1909
§ Roberts, Nicholas J., Central Mine, Broken Hill, N.S.W.	1913
Rodda, E. J. J., "Ensanada," Murdock-street, Cremorne, Sydney, N.S.W.	1907
Rodda, Stanley N., Working Men's College, Melbourne, Vic.	1910
Rolfe, Richard A., 53 West-street, North Sydney, N.S.W.	1915
Rowe, F. Walton, Oroya Links Ltd., Kalgoorlie, W.A.	1914
Rusha, Richard A., c/o Institution of Mining & Metallurgy, 225 City-road, London, E.C. 1.	1912
Russell, Murray, c/o Dept. of Mines, Brisbane, Q.	1912
Ryan, Cecil G., Bradshaw's Creek, Tas.	1908
Sander, George E., c/o The Rhodesian Lead and Zinc Syndicate Ltd., Broken Hill, Rhodesia, South Africa	1910
§Sarvaas, Johan, Working Men's College, Melbourne, Vic.	1902
Saunders, George J., Technical College, Ipswich, Q.	1904
Savage, A. E., "Nangeela," Point-road, Woolwich, Sydney, N.S.W.	1911
Schlapp, H. H., Collins House, 360 Collins-street, Melbourne, Vic.	F.
Scoble, E. J., Station-road, Waihi, N.Z.	1917
Scott, Malcolm B., c/o Mrs. Moore, 78 Ward-street, Nth. Adelaide, S.A.	1909
Shaw, J. B., Metal Manufacturers' Ltd., Port Kembla, N.S.W.	1910
Shepherd, H. Frank, Hauraki Mines, Coromandel, N.Z.	1911
¶§ Skeats, Professor E. W., The University, Melbourne, Vic.	1905
§Slee, Richard T., Broken Hill Pty. Ltd., Broken Hill, N.S.W.	1902
§Slessor, Robt., Mining Dept., Arnhold Bros. and Co., Changsha, Hunan Province, China	1905
Smith, C. E. Kennedy, State Battery, Ora Banda, W.A.	1915
Smith, C. Lonsdale, "Darnick," Larkin-street, North Sydney, N.S.W.	1905
§Smith, F. Voss, Block 14 Co., Broken Hill, N.S.W.	1909
Smith, J. D. Audley, Box 1557, P.O., Sydney, N.S.W.	1910
Somerset, H. St. J., Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1898

Names and Addresses.	Date of Election.
Stephen, James F., "Rydalmount," Webb's-avenue, Ashfield, Sydney, N.S.W.	1918
Stephenson, Clement T., 39 Wrixon-street, East Kew, Vic.	1912
Stewart, Farquhar, Walsh-street, Reefton, Westland, N.Z.	1910
¶§Stewart, Hector, "Solai," Wagin, W.A.	1902
Stillwell, F. L., 44 Elphin-grove, Hawthorn, Vic.	1921
Stirling, J., Burnie, Tas.	F.
Stokes, C. H., 9 Harvest Terrace, W. Perth, W.A.	1910
Stokes, Reginald H., 48 Elizabeth-street, Sydney, N.S.W.	1911
Stutterd, Louis N., Post-office, Port Darwin, Northern Territory	1919
Sussmilch, C. A., Technical College, Newcastle. N.S.W.	1904
Sutherland, John W., Golden Horseshoe Estates Ltd., Boulder, W.A.	1909
Sutherland, Wm. G., Fremantle Smelting Works, Fremantle. W.A.	1909
Taylor, E. Hogan, "Wyuna," 30 Carlisle-street, St. Kilda, Vic.	1913
Thomas, Harry A., Carisbrook, Vic.	1913
§Thompson, Percival F., The University, Melbourne, Vic.	1922
Trenerry, Edw. Jos., Moonta Mines, S.A.	1911
Turnbull, T. A., Nelson, N.Z.	F.
§Turri, G. G., The Rialto, Collins-street, Melbourne, Vic.	F.
Turrell, Samuel George, 13 Boulder-road, Kalgoorlie, W.A.	1921
Ullman, A. T., Clarendon Hotel, Hunter-street, Newcastle, N.S.W.	1914
Ulrich, Gerard A., c/o Inst. of Mining and Metallurgy, 225 City-road, London, E.C. 1	1911
Vaudeau, H. A., Inspector of Mines' Office, Upper Burnie, Tas.	1922
Waddell, W. A., "Hiawatha," Esplanade, Cairns, Q.	F.
§Wainwright, W. E., Broken Hill South Ltd., 360 Collins-street, Melbourne, Vic.	1902
§Wallmann, H. P., 26 Vautier-street, Elwood, Vic.	1910
Walls, Goiders, Devonport West, Tas.	1920
§Walsh, Peter Jos., c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1920

Names and Addresses.	Date of Election.
Walton, Robert W., 324 Bondi-road, Bondi, N.S.W. ..	1911
Ward, L. Keith, Department of Mines, Adelaide, S.A. ..	1918
§Waterhouse, Leslie V., 58 Pitt-street, Sydney, N.S.W. ..	1915
§Waters, Professor D. B., The University, Dunedin, N.Z.	1904
Watson, James T., Chillagoe Ltd., Mt. Mulligan, via Cairns, Q.	1911
Weigall, A. R., c/o Seoul Mining Co., Suan Mine, Hol Kol, Whang Hai Province, Korea	1910
Weigall, H. S., Taio G.M. Co. Ltd., c/o Post Office, Yabe, Fukuokaken, Kiushiu, Japan	1912
Weir, George, 360 Collins-street, Melbourne, Vic.	1902
Westcott, L. A., Lithgow Iron and Steel Works, Lithgow, N.S.W.	1907
§White, Edmond A., Electrolytic Refining Co., Port Kembla, N.S.W.	1900
White, Francis P., Goonengerry, via Mullumbimby, North Coast, N.S.W.	1913
§White, Norman F., "Warrandaroo," Longreach, Q. ..	1899
Whitfeld, Professor H. E., The University, Perth, W.A.	1914
Whitley, Alexander, Inspector of Mines, Dunedin, N.Z. ..	1911
Wilkins, Laurence G., Chemical Laboratory, Broken Hill Proprietary Co. Ltd., Newcastle, N.S.W.	1913
§Williams, Luke, Claremont Estate, Moonah, Tas. ..	1894
§Williams, Owen B., c/o North Mount Farrell Co., N.L., Tullah, Tas.	1920
Williams, Thomas H., Leonora, W.A.	1910
Wilson, A. Laurence, New Ravenswood Ltd., Ravenswood, Q.	1909
Wilson, Douglas de C., c/o Junction North Mine, Broken Hill, N.S.W.	1920
Wilson, F. Grey, Tingha Tin N.L., Tingha, N.S.W. ..	1916
Wilson, Harry, 31 Queen street, Melbourne, Vic. .	1914
Wilson, William T. S., Pæroa, Auckland, N.Z.	1911
Winter, Arthur Sydney, Cobar, N.S.W...	1918
§Woodward, Oliver H., c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1919
Worsley, Henry E., Broken Hill Pty. Ltd., Broken Hill, N.S.W.	1910
§Wright, Geo. W., National Portland Cement Co., Maria Island, Tas.	1906
Young, Gerald W., "Rona," Toorak-road, Toorak, Vic. ..	1909

ASSOCIATE MEMBERS.

Names and Addresses.	Date of Election.
Adcock, Garnet I., Gosford, N.S.W.	1919
Alford, L. C., "Oakhampton," Wellington, N.Z.	1910
Anderson, A. F. S., The Rectory, New Norfolk, Tas.	1914
Anderson, William T., "Clifton," Ben Boyd-road, Neutral Bay, N.S.W.	1920
Armstrong, Andrew S., Chillagoe, Q.	1912
Atkinson, J., Emmaville, N.S.W.	1907
Auld, James B., Crushington, N.Z.	1922
Baldwin, Percy H., Chelmsford-avenue, Wooloowin, Brisbane, Q.	1910
Banks, John H. G., Waihi G.M. Co., Waihi, N.Z.	1910
Barkley, Ernest, Mount Lyell Batch, Gormanston, Tas.	1911
Beckwith, Ralf N., c/o Sulphide Corporation Ltd., Port Pirie, S.A.	1910
Berriman, R. V., Technical School, Port Pirie, S.A.	1920
Black, Arnold B., c/o Broken Hill South Ltd., Broken Hill, N.S.W.	1912
Boyd, Eric E. G., Mount Morgan, Q.	1921
Boydell, W. G. B., c/o Seoul Mining Co., Hol Kol Mine, Whang Hai Do, Chosen, Korea	1911
Broad, Wilfred R., c/o British Australian Broken Hill Ltd., Broken Hill, N.S.W.	1920
Brown, G. Twentyman, "Glenfern," Mathoura-road, Toorak, Vic.	1910
§Burbidge, Percy, 20 Avoca-street, South Yarra, Vic.	1913
Caithness, W. R., c/o Fraser and Chalmers Ltd., 257 Clarence-street, Sydney, N.S.W.	1909
Carless, Noel, Walney House, Waihi, N.Z.	1913
§Carroll, Hubert H., c/o Broken Hill South Ltd., Broken Hill, N.S.W.	1920
Casey, Richard G., 125 William-street, Melbourne, Vic.	1920
Christian, Arthur J., The Ranch, Oxide-street, Broken Hill, N.S.W.	1910
Clapp, L. P., 76 Chloride-street, Broken Hill, N.S.W.	1910
Clark, G. C. Lindesay, 6 Brookville-road, Toorak, Vic.	1923

Names and Addresses.	Date of Election.
Clark, Archie S., c/o Amalgamated Zinc (De Bavay's) Ltd., Broken Hill, N.S.W.	1910
Cole, William H., Waihi, N.Z.	1912
Collison, Arthur G., Eagle Chambers, King William-street, Adelaide, S.A.	1902
Combe, A. D., Geological Survey, Entebbe, Uganda, via Mombassa, British East Africa	1922
Cooke, Herbert R., N.Z. Express Coy's. Building, 12 Fort-street, Auckland, N.Z.	1911
Corstophan, Valentine A., c/o Electrolytic Zinc Co. of Australasia Ltd., Risdon-road, Hobart, Tas.	1915
Court, A. Lacy, c/o Central Mine, Broken Hill, N.S.W.	1913
§Courtney, Guy, c/o Sulphide Corporation Ltd., Boolaroo, N.S.W.	1912
Crace, Melville C., Broken Hill South Ltd., Broken Hill, N.S.W.	1911
Craig, James H., Albion Chambers, Bendigo, Vic.	1913
Crawford, Hugh, School of Mines, Thames, N.Z.	1921
Cropp, Walter H. J., Mount Bischoff Tin Co., Waratah, Tas.	1922
Danks, Aaron T., 391 Bourke-street, Melbourne, Vic.	1905
Dod, H. Charlton, Evans-street, Belmont, Geelong, Vic.	1914
§Dow, Edmund B., c/o Central Mine, Broken Hill, N.S.W.	1910
Dow, John, Golden Horseshoe Estates Co. Ltd., Boulder, W.A.	1910
§Drain, Percy J., Kitchener-road, Artarmon, Sydney, N.S.W.	1910
Dunn, John Alexander, c/o Geological Survey of India, Calcutta, India	1921
Dunstan, Basil G., c/o Australian Slate Quarries, Willunga, S.A.	1911
§Edwards, Ed., "Glenesk," Burke-road, Canterbury, Vic.	1907
Elford, Chas., P.O. Box No. 11, Chillagoe, North Q.	1915
Fabrikant, Isodore B., 50 Erskine-street, North Melbourne, Vic.	1922
Flockart, Colin P., 58 Broadway, East Camberwell, Vic.	1912
Fox, James Michael, North Broken Hill Ltd., Broken Hill, N.S.W.	1910
§Foxall, John S., Northwood-road, Longueville, Sydney, N.S.W.	1912

Names and Addresses.	Date of Election.
Francis, Eben T., c/o Amalgamated Zinc (De Bavay's) Ltd., Broken Hill, N.S.W.	1922
Fry, Arthur E., c/o Zinc Corporation Ltd., Broken Hill, N.S.W.	1919
§Fry, Sidney, Reefton, N.Z.	1907
Gabriel, G. Escot, 95 Alma-road, East St. Kilda, Vic.	1920
Game, George B., Eildon Weir, via Alexandra, Vic.	1918
Garnett, Robert K., c/o British Australian Broken Hill Co. Ltd., Broken Hill, N.S.W.	1920
Gluyas, B. C., Broken Hill Pty. Ltd. Steel Works, Newcastle, N.S.W.	1913
Gray, William, Anglo-Persian Oil Co. Ltd., Papua	1921
Greening, Walter, Te Aroha, N.Z.	1911
Greenlees, Alan D., Amalgamated Zinc (De Bavay's) Ltd., Broken Hill, N.S.W.	1910
Grieve, William H., c/o William Adams and Co., 175 Clarence-street, Sydney, N.S.W.	1911
Grosse, William J., 63 Patton-street, South Broken Hill, N.S.W.	1913
Guppy, Clarence H. R., Nowra, N.S.W.	1911
Hart, William, Mount Lyell Mining and Railway Co. Ltd., Gormanstown, Tas.	1919
Harvey, Phillip, School of Mines, Charters Towers, Q.	1913
Heycock, C. R., Ohai, via Nightcap, N.Z.	1914
Hooke, Arthur W., Forum River (Nigeria) Tin Co. Ltd., Naraguta, Northern Nigeria, West Africa	1911
§Hookings, Herbert J., Broken Hill Pty. Ltd., Broken Hill, N.S.W.	1911
Hooper, Francis H., Block 14 Mine, Broken Hill, N.S.W.	1920
Hooper, Thomas Henry, Mount Royal Copper Mines, Tottenham, N.S.W.	1919
§Hope, G. B., "Carrical." Hermitage-rd., Newtown, Geelong, Vic.	1911
Horsburgh, W. W., Hampden-Cloncurry Copper Mines Ltd., Kuridala, via Cloncurry, North Q.	1911
Hughes, Eric W., School of Mines, Adelaide, S.A.	1921
§Hunter, George M'I., c/o Electrolytic Zinc Co. of Australasia Ltd., Risdon, Hobart, Tas.	1910
Irvine, C. L., c/o Mount Marshall Farmers Co., Bencubbin, W.A.	1913

Names and Addresses.	Date of Election.
Jones, Vincent N., Beaudesert Shire Council, Beaudesert, Q.	1910
§Junner, Norman R., c/o Gold Coast Geological Survey, Colonial Office, Downing-street, Westminster, London	1920
Kneebone, Christopher Stephen, Woodville-road, Woodville, S.A.	1921
Langford, Walter G., Anglo-Persian Oil Co. Popo, Papua	1917
Lawn, Charles H., Opunake-road, Stratford, Taranaki, N.Z.	1911
Lawrance, Dudley S., Fifield, via Trundle, N.S.W.	1921
Leevers, William T , Great Boulder Perseverance G.M. Co., Fimiston, W.A.	1910
Le Souef, Sebert G., Royal Mint, Melbourne, Vic.	1910
Lewis, Keith B., c/o Aust. Raub G.M. Co., Raub, Pahang, Fed. Malay States	1915
Linz, Carl J., T. & G. Building, Sydney, N.S.W.	1920
§MacGruer, Donald C., c/o Phosphate Co-Operative Co. of Australasia, 440 Lit. Collins-street, Melbourne, Vic.	1915
McKay, James, Box 55, P.O., Greymouth, N.Z.	1911
Melhuish, Theodore D'A., Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1923
Montgomery, John N., Upoia, Papua	1914
§Moore, Malcolm S., 31 Queen-street, Melbourne, Vic.	1912
§Morgan, R. J., Spassky, Zavod, Akmolinsk, Siberia	1911
§Murrie, Jack E. P., 44 Goode-road, Port Pirie, S.A.	1923
§Nevett, Ralph D., Thomas-lane, off Chloride-street, Broken Hill, N.S W.	1912
Newbigin, William J., 175 Clarence-street, Sydney, N.S.W.	1913
Nicholas, Frank H., c/o F. C. T. Nicholas, Metal Exchange, Collins Court, Melbourne, Vic.	1913
Noall, John, c/o Pahang Consolidated Co. Ltd., Sungei Lembing, Kuantan, Singapore, Fed. Malay States	1921
O'Nians. P. H., The Mexican Corporation, S.A. Aire Libre Puebla, Mexico	1922
Oliver, Robert B., c/o White & Hosler, Broken Hill, N.S.W.	1919
Osborne, George W., c/o Yukon Dredging Co., Kuala Lumpur, Fed. Malay States	1913
§Ower, Leslie H., Belize, British Honduras	1910
§Pearson, H. F., Zinc Corporation Ltd., Broken Hill, N.S.W.	1920
Pethebridge, Harry V., Main-road, Moonah, Hobart, Tas.	1920

Names and Addresses.	Date of Election.
Rabling, Harold, 69 Primrose-street, Moonee Ponds, Vic.	1914
Rain, Allan D. M., Braden Copper Mines, Rancagua, Chili, South America	1911
Ramsden, Frederick V., c/o J. W. Dudgeon, Scottish House, William-street, Melbourne, Vic.	1921
Reid, Geo. D., Royal Mail Hotel, Stradbroke, Vic.	1916
Robertson, Herbert James, Bibiani Gold Mines, Gold Coast, West Africa	1920
Robson, H. Carmichael, "Silverdale," Penzance, Cornwall, England	1913
Roberg, Frederick A., B.H.P. Block 10 Co. Ltd., Broken Hill, N.S.W.	1922
Rogers, George, c/o Burma Mines Ltd., Namtu, Northern Shan States, Burma	1913
Rufus, F. A., Block 10 Mine, Broken Hill, N.S.W.	1920
Ryan, D. L., Fremantle Trading Co., Fremantle, W.A.	1917
Salter, John W., Inspector of Mines, Broken Hill, N.S.W.	1912
Sampson, Thomas, c/o Automatic Engineering Co., Hiles-street, Alexandria, Sydney, N.S.W.	1919
Sampson, John L., c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1921
Seale, Harry V., Junction North Mine, Broken Hill, N.S.W.	1910
§Sewell, R. M., School of Mines & Industries, Stawell, Vic.	1913
Shaw, John F., Block 14 Mine, Broken Hill, N.S.W.	1912
Simpson, James, Crown Law Dept., Perth, W.A.	1910
Sligo, A. D., Zeehan, Tas.	1911
Smith, Joseph F., c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1923
Smith, R. Dermer, Hampden-Cloncurry Copper Mines Ltd., Trekelano, via Duchess, North Q.	1908
§Smith, Walter G. A., Mine Office, Lionsville, via Grafton, N.S.W.	1912
Stafford, H. R., c/o Burma Corporation Ltd., Nam Tu, N.S.S., Burma	1923
Stephen, Fenton W., Broken Hill Pty. Co., Broken Hill, N.S.W.	1910
Stephenson, Godfrey, "Kaioni," Point-road, Northwood, Sydney, N.S.W.	1917

Names and Addresses.	Date of Election.
Stephenson, Herbert, "Beaumaris," Fletcher-street, Bondi, Sydney, N.S.W.	1910
Thomas, Frank N., Broken Hill South Ltd., Broken Hill, N.S.W.	1911
Thompson, Charles G. L., 85 Rose-street, Armadale, Vic.	1921
Thomson, F. D., c/o Alden's Successors Ltd., Penang, Straits Settlement	1908
Tinworth, Revel, "Rochester," Regent-street, Ryde, Sydney, N.S.W.	1921
Topp, A. A., 12 Cromwell road, Hawksburn, Vic.	1909
Truscott, F. J., Block 10 Mine, Broken Hill, N.S.W.	1922
Twine, George J., Queensland Cement and Lime Co., Darra, Q.	1914
Vaughan, Rupert M., Milton Bryan Rectory, Woburn, Beds., England	1911
§Warren, P. Hampton, Broken Hill South Ltd., Broken Hill, N.S.W.	1910
§Watkins, Augustus E., St. Aubyn-street, New Plymouth, Taranaki, N.Z.	1911
Watson, Herbert, c/o Amalgamated Zinc (De Bavay's) Ltd., Broken Hill, N.S.W.	1911
White, Eric H. D., School of Mines, Bairnsdale, Vic.	1921
White, Fredk. G. (White & Hosier), Broken Hill, N.S.W.	1911
Whitehead, Edwin A., "Aballaba," 6 North Terrace, Kensington Gardens, S.A.	1902
Whiteman, W, D., British Australian Broken Hill Ltd., Broken Hill, N.S.W.	1913
Willan, Thos. L., Geological Survey Office, Department of Mines, Sydney, N.S.W.	1921
Williams, Arthur, Oxide Creek Ltd., Mt. Gordon Mine, via Cloncurry, Q.	1915
Williams, Oswald M., Inspector of Mines, Mareeba, Nth. Q.	1919
Wilson, Wilfred A., P.O. Box 73, Whangarei, N.Z.	1911
Winters, R. J., c/o Geological Survey, Entebbe, Uganda, British East Africa	1914
Zeugofsge, F. C., 288 Patton-street, South Broken Hill, N.S.W.	1910

JUNIORS.

Names and Addresses	Date of Election.
Allsop, John R., 14 Queen-street, Solomontown, S.A.	1923
Ellis, Leonard V., c/o Auckland Gas Co., Beaumont-street, Auckland, N.Z.	1923
Goddard, Clarence L. B., 26 Commercial-road, North Hobart, Tas.	1923
Goodyear, Robert T., 181 Chloride-street, Broken Hill, N.S.W.	1922
Schache, Carl A., The Armidale School, Armidale, N.S.W.	1921

STUDENTS.

Anderson, Archibald G., 4 Hunter-street, Newcastle, N.S.W.	1913
Anderson, E. S., c/o R. E. Chapman, Bank of New South Wales, Ballarat, Vic.	1912
Barratt, Alfred H. D., Technical College, Broken Hill, N.S.W.	1923
Beal, Dick, British-Australian Broken Hill Ltd., Broken Hill, N.S.W.	1922
Blunden, Leslie E., c/o Wildridge & Sinclair, 228 Elizabeth-street, Brisbane, Q.	1910
Brooke, Malcolm A., St. Andrews-street, North Walkerville, S.A.	1923
Buckley, Horace P., 59 Sinclair-street, Woolstonecraft, N.S.W.	1923
Burns, Felix G., Terminus Hotel, Williamsford, Tas.	1914
Cerutty, W. E., 415 Chapple-street, Broken Hill, N.S.W.	1922
Colman, Walter R. G., 502 William-street, Broken Hill, N.S.W.	1920
Corrie, Colin C., Drafting Office, Dept. of Works and Railways, Treasury Gardens, Melbourne, Vic.	1911
Dale, Roger T. H., c/o Taratu Coal Mines, Lovell's Flat, Otago, N.Z.	1920
Elford, Harold Stewart, 19 Beulah-road Extension, Norwood, Adelaide, S.A.	1922
Eriksen, Alfred L., 336 Thomas-street, Broken Hill, N.S.W.	1922
Fisher, George R., 4th-avenue, Joslin, Adelaide, S.A.	1923
Fisher, Ivan, 11 Stanley-grove, Canterbury, Vic.	1921

Names and Addresses.	Date of Election.
Forster, John A. B., Pressland House, Melville-street, Hobart, Tas.	1922
Fraser, Ernest H., c/o Electrolytic Zinc Co. of Australasia, Zeehan, Tas.	1911
Gibbons, L., P. J., W.A. School of Mines, Kalgoorlie, W.A.	1920
Giles, George R., c/o Swinburne Technical College, Hawthorn, Vic.	1923
Goldrick, M. R., c/o Mrs. Kruchowski, Eyre-street North, South Broken Hill, N.S.W.	1921
Goode, Kenneth B., c/o Matthew Goode and Co., 18 Silk-street, London, E.C.2	1913
Green, Harold L., c/o W. Green, Union S.S. Co. (N.Z.) Ltd., Dunedin, N.Z.	1920
Gross, Klem. B., 422 William-street, Broken Hill, N.S.W.	1922
Hill, James, Main Roads Board, Cairns, Nth. Q.	1907
Hooper, Raymond P., c/o Broken Hill South Ltd., Broken Hill, N.S.W.	1922
Hopkins, John, c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.	1920
Horsley, Ray D., Stannary Hills, North Q.	1910
Howlett, Hugh B., 65 Macgregor-street, East Malvern, Vic.	1921
Jefferson, F. M., c/o Zinc Corporation Ltd., Broken Hill, N.S.W.	1923
Kelly, A. H., Croas-street, Mayfield, N.S.W.	1913
Kerr, Frederick T., Central Mine, Broken Hill, N.S.W.	1918
Lambert, Charles E., "Mildred," 9th-avenue, Campsie, N.S.W.	1923
M'Donald, Allen C., 49 York-road, Pirie West, Port Pirie, S.A.	1923
M'Leod, A. A., Lucknow, Bairnsdale, Vic.	1921
M'Mahon, Gregan T. H., 150a Elizabeth-street, Sydney, N.S.W.	1921
Meikle, Robert A. R., Dee-street, Mount Morgan, Q.	1920
Osborne, Wilfred St. C., 15 Raventhorpe-avenue, Millswood Estate, Adelaide, S.A.	1923
Paterson, Robt. B., c/o Morning Star Gold Mines, Wood's Point, Vic.	1922
Plant, G. H. D., "Thornburgh," Charters Towers, Q.	1910

Names and Addresses.	Date of Election.
Pyke, Henry G., Cambridge-street, Penshurst, Sydney, N.S.W.	1923
Russell, Reuben S., Manifold-street, Camperdown, Vic.	1923
Sheil, William G., 510 Rathdown-street, North Carlton, Vic.	1923
Smith, Eugene, 82 Victoria road, Auburn, Vic.	1913
Stacey, Kingsley H., Broken Hill Associated Smelters, Port Pirie, S.A.	1923
Symonds, G. B. L., 174 Wolfram-street, Broken Hill, N.S.W.	1916
Taylor, Augustus S., Geological Survey, Entebbe, Uganda, British East Africa	1908
Tobe, Theodore W., Aust. Fertilizers Pty. Ltd., Port Kembla. N.S.W.	1921
Turner, William A., c/o State Electricity Commission, Yallourn, Vic.	1914
Uren, Wm. T., Salmon Mine, Dolomite, Q.	1909
Veitch, Neil A., Hampden-Cloncurry Copper Mines Ltd., Kuridala, via Cloncurry, North Q.	1911
Watts, Frank, Returned Soldiers' Hostel, Broken Hill, N.S.W.	1921
White, James M., c/o Malaya Tin Corporation Ltd., Puket, West Siam	1919
Wicks, Cecil W., 3 Anderson-street, Marrickville, Sydney, N.S.W.	1921
Williams, W. H. (jun.), Assistant Inspector of Mines, Rockhampton, Q.	1910
Wishart, J. W., H.M.A.S. *Torrens*, c/o G.P.O., Sydney, N.S.W.	1914
Yates, Donald, Mayfield, Newcastle, N.S.W.	1914

PROCEEDINGS

OF THE

AUSTRALASIAN

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(INCORPORATED).

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No. 51.

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Australasian Institute of Mining and Metallurgy

(INCORPORATED).

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Representing South Australia—

H. Lipson Hancock

Representing New Zealand—

D. B. Waters

Representing New South Wales—

G H. Blakemore
H. A. Evans
C. C. Freeman
F. Danvers Power
F. Voss Smith

Representing Victoria—

Geo. H. Broome
H. Herman
E. W. Skeats

Representing Western Australia—

Richard Hamilton
J. W. Sutherland

Representiug Tasmania—

H. W. Gepp
R. M. Murray

Ex officio Member of Council—

G. C. Klug (Past President).

Hon. Treasurer:

A. S. Kenyon.

Bankers:

Commonwealth Bank of Australia, Collins Street, Melbourne, Vic.

Secretary:

D. L. Stirling, Collins Court, 374 Little Collins Street, Melbourne, Vic.

BRANCH SECRETARIES:

Broken Hill - C. H. J. Clayton, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie - S. G. Turrell, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan - B. G. Patterson, Mount Morgan, Q.
Port Pirie - G. Dey, c/o Broken Hill Associated Smelters, Port Pirie.
Sydney - F. Danvers Power, 31 Hunter Street, Sydney, N.S.W.

LOCAL CORRESPONDENTS:

South Australia - H. W. Gartrell, The University, Adelaide.
New Zealand - D. B. Waters, The University, Dunedin.
Tasmania - Russell M. Murray, Queenstown.
North Queensland Murray Russell, Department of Mines, Brisbane.
Papua - - W. G. Langford, c/o Anglo-Persian Oil Co., Popo, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

Collins Court, 374 Little Collins Street, Melbourne, Victoria.

Institute Matters.

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MINUTES OF MEETINGS.

OF THE EXECUTIVE COMMITTEE.

(Summary.)

JULY 30TH, 1923—1 P.M.

The Secretary's report was presented, and accounts to the amount of £13 11s. 2d. passed for payment.

Report of meeting of the Broken Hill Branch held on 19th April was submitted and received.

A paper entitled "Notes on the Lindsay Reward Tin Mine, Western Tasmania," by D. V. Allen, was referred to the Publication Committee.

Resolved that a letter be forwarded to Mr. James Hebbard, manager of the Central mine, Broken Hill, expressing sympathy on account of the fire at that mine.

The nomination of Mr. R. H. Berriman as an Associate Member was approved for ballot.

Messrs. Gilbert Rigg and Wm. Baragwanath were nominated to represent the Institute on the Paints and Paint Oils Sectional Committee of the Commonwealth Engineering Standards Association.

The nomination of Messrs. J. B. Lewis and W. R. Caithness as the Institute's representatives on the Electrical Standards Section of the Commonwealth Engineering Standards Association was endorsed.

A cable from the Secretary, I.M.M. (London), reporting that Mr. Hugh F. Marriott was visiting Australia early in August, and desired to meet the Council of the Aus. I.M.M., was received, and arrangements made for the interview.

The following clause, which it was proposed should be added to the Constitution of the Federal Engineering Council, was approved :—

> "Clause 11, Amendment to the Constitution.—Amendments of the Constitution may be proposed (*a*) by resolution of the Council, (*b*) by resolution of the Council of one or more of the constituent bodies. Such amendments shall be submitted to the Councils of all the constituent bodies, and shall be adopted only subject to their unanimous approval."

Messrs. G. C. Klug and C. F. Courtney were nominated to represent the Aus. I.M.M. at the World Power Conference, to be held in London in 1924.

AUGUST 27TH, 1923—1 P.M.

Minutes of meeting held on 30th July were read and confirmed.

The Secretary's report was presented, and accounts to the amount of £72 13s. 7d., including salary and rent (July and August), £66 13s. 4d., and sundries, £8 0s. 3d., were passed for payment.

Nomination of Mr. Leonard Banks Haney as a Junior was approved for ballot.

Mr. Maurice A. E. Mawby was admitted as a Student.

As a result of ballot by members of Council, Mr. R. H. Berriman was declared elected an Associate Member.

A report of a meeting of the Broken Hill Branch held on 21st August was submitted and received. It was reported that copies of matter submitted to the Pan-Pacific Congress desired by the Branch were not procurable.

Resolved that the Council had no objection to the question of preliminary general training that candidates must possess before qualifying for admission to the bodies constituting the Federal Engineering Council being discussed by that Council.

Report of a special meeting of the Main Committee of the Commonwealth Engineering Standards Association was presented and received.

Reports were also received from the Federal Engineering Council as follows :—

(1) Annual report for term ending 30th June, 1923.

(2) Report of fourth meeting of Federal Engineering Council held on 31st July, 1923. This report included the following recommendations in regard to the proposed establishment of an employment bureau :—

  (1) That a Central Employment Bureau common to the three constituent bodies be instituted.

  (2) That an Employment Register be opened at headquarters, the secretaries of the constituent bodies supplying the General Secretary of the Federal Council with the names of unemployed members, together with other requisite information.

  (3) That each constituent body keep a list of its own members desiring employeemnt.

  (4) That headquarters circularize the principal employers, informing them of the institution of the Employment Bureau and of its scope.

  (5) That, in order to meet expenses incurred, each constituent body pay a sum proportionate to its total membership, the total annual expenditure not to exceed £50.

Resolved that fuller information be obtained regarding the proposed World Power Conference, 1924, and that the Secretary I.M.M. be communicated with in the matter.

It was decided (1) that copies of the report of Mr. H. F. Marriott's interview with the Executive Committee on 16th August be forwarded to all members of Council, with the request that opinions thereon be forwarded by 10th September ; (2) that opinions received be summarized, and a recommendation on the subject prepared.

Resolved that the Institute enter its protest against the proposal to remove the Victorian Geological Museum and Library from its present admirable location to the Exhibition Building.

SEPTEMBER 24TH, 1923—1 P.M.

Minutes of the previous meeting were confirmed.

The Secretary's report was presented, and accounts to the amount of £60, including printing, £46, typing reports, £5, and petty cash, £8, were passed for payment.

As a result of ballot by members of Council, Mr. Leonard Banks Haney was declared elected a Junior Member.

The following applications for admission to the Institute were approved for ballot:—As Associate Member—Mr. John Hopkins; as Juniors—Messrs. John Arthur Blackett Forster and Alan Trevor Davies.

The following were admitted as Students:—Messrs. Adolf Theodor Ahlston, Maxwell Ernest Playford, and Henry Kum Yuon.

Resignations of Messrs. H. J. Robertson and B. C. Gluyas were accepted.

Nominations of Messrs. Walter G. A. Smith and Herbert S. Roberts were deferred for fuller information.

It was reported that replies from all members of Council on the proposal to form an Empire Council of Mining and Metallurgy were favourable, and that Mr. H. F. Marriott had been so informed.

It was resolved that appointments to the proposed Empire Council of Mining and Metallurgy be made in February, 1924.

It was also resolved that the preparation of a paper for the proposed Empire Mining and Metallurgical Congress, referred to in the memorandum from the Institution of Mining and Metallurgy and the Institution of Mining Engineers (a copy of which was forwarded to members of Council on 24th August), be left in the hands of the Publication Committee.

A paper by Mr. H. A. Evans on "Lead Smelting at the Sulphide Corporation Ltd.'s Cockle Creek Works" was submitted and referred to the Publication Committee.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

The following is a list of members admitted to the Institute in July and August, 1923:—

Members.

Names and Addresses.

HEMINGWAY, ARTHUR JOHN, 50 Goode-street, Port Pirie, S.A.
HUGHES, JOHN CLARENCE, 8 Thomas-street, Port Pirie, S.A.
JONES, CUTHBERT EDWARD REES, c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.

Associate Members.

BERRIMAN, ROBERT HARROLD, c/o Electrolytic Zinc Co. of Australasia Ltd., Risdon-road, Hobart, Tas.
CLARK, GORDON COLVIN LINDESAY, 6 Brookville-road, Toorak, Vic.
MELHUISH, THEODORE D'AVELINE, c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.
MURRIE, JACK ESDALE PARR, 44 Goode-street, Port Pirie, S.A.
SMITH, JOSEPH FREDERICK, c/o Broken Hill Associated Smelters Ltd., Port Pirie, S.A.

Juniors.

ALLSOP, JOHN RICHARD, 14 Queen-street, Solomontown, S.A.
GODDARD, CLARENCE LEWIS BOWDEN, 26 Commercial-road, North Hobart, Tas.

Student.

MAWBY, MAURICE ALLAN EDGAR, 475 Wyman-street, Broken Hill, N.S.W.

COMMONWEALTH ENGINEERING STANDARDS ASSOCIATION.

Messrs. Gilbert Rigg and W. Baragwanath have been nominated to represent the Institute on the Paints and Paint Oils Sectional Committee of the Commonwealth Engineering Standards Association, and Messrs. James B. Lewis and W. R. Caithness on the Electrical Standards Section.

FEDERAL ENGINEERING COUNCIL.

The following clause, which it is proposed should be added to the Constitution of the Federal Engineering Council, has been approved by the Council of the Aus. I.M.M. :—

> "Clause 11—Amendment to the Constitution.—Amendments to the Constitution may be proposed (*a*) by resolution of the Council, (*b*) by resolution of the Council of one or more of the constituent bodies. Such amendments shall be submitted to the Councils of all the constituent bodies, and shall be adopted only subject to their unanimous approval."

WORLD POWER CONFERENCE.

Messrs. G. C. Klug and C. F. Courtney have been nominated to represent the Aus. I.M.M. at the World Power Conference to be held in London in 1924.

CENTRAL EMPLOYMENT BUREAU.

The following recommendations have been adopted by the Federal Engineering Council :—

(1) That a Central Employment Register common to the three constituent bodies be instituted.

(2) That an Employment Register be opened at headquarters, the secretaries of the constituent bodies supplying the General Secretary of the Federal Engineering Council with the names of unemployed members, together with other requisite information.

(3) That each constituent body keep a list of its own members desiring employment.

(4) That headquarters circularize the principal employers, informing them of the institution of the Employment Bureau and its scope.

(5) That, in order to meet expenses incurred, each constituent body pay a sum proportionate to its total membership, the total annual expenditure not to exceed £50.

Members of the Aus. I.M.M. to whom clause 3 above applies are requested to supply the Secretary, Aus. I.M.M., with full particulars of education and experience, and to indicate the kind of employment they particularly desire.

EMPIRE COUNCIL OF MINING AND METALLURGY.

Proposals in regard to the formation of an Empire Council of Mining and Metallurgy have been framed by the Institution of Mining and Metallurgy (London) and the Institution of Mining Engineers (London). These proposals are embodied in the following memorandum from the above institutions :—

"The arrangement for intimate co-operation between the Institution of Mining Engineers and the Institution of Mining and Metallurgy, which has been effective for about a year, has already been attended by results which must benefit the mining and metallurgical professions and industries—results which will become increasingly evident and important as time passes.

"For many years it has been evident that the absence of unity and close co-operation between the various mining and metallurgical institutions at home and in the Dominions is a source of weakness and detrimental to the interests of the professions which are identified with those of the industries and of the Empire.

"There are numerous institutions in the British Isles and in the Dominions whose members are, in varying degrees, practising mining and metallurgical engineers.

"In addition to the Institution of Mining and Metallurgy and the seven federated institutes constituting the Institution of Mining Engineers, there are the Institution of Petroleum Technologists, the Iron and Steel Institute, the Institute of Metals, and the South Wales Institute of Engineers in the British Isles ; and there are in the Dominions the Canadian Institute of Mining and Metallurgy, the Australasian Insitute of Mining and Metallurgy (until recently the Canadian Mining Institute and the Australasian Institute of Mining Engineers respectively), the Chemical, Metallurgical and Mining Society of South Africa, the South African Society of Engineers, the Mining and Geological Institute of India, and possibly others not included in this list.

"Much thought has been given to the question, and we are convinced that the interests of the mining and metallurgical professions and industries would be materially advanced by the formation of an Empire Council of Mining and Metallurgical Institutions, whose principal functions would be :—

"(*a*) To create and maintain throughout the Empire a high standard of technical efficiency and professional conduct.

"(*b*) To convene at regular intervals an Empire Mining Congress, the first to be held in connection with the British Empire Exhibition in London in 1924, and subsequent congresses to be held in the Dominions.

"(*c*) To establish (if found, after discussion by the Council and the co-operating Institutions, to be necessary or desirable) a Register of *British* Mining and Metallurgical Engineers.

"(*d*) To serve as an organ of intercommunication between the co-operating institutions and to promote the interests of the professions and industries.

"(Note.—The Constitution of the Council might consist in the first instance of, say, two representatives of each of the co-operating institutions.)

"The following suggestions are submitted for consideration in the first instance by the Joint Advisory Committee, and later by the Councils of the two institutions:—

"(1) That steps be taken to unite the mining and metallurgical engineering professions in an 'Empire Council of Mining and Metallurgical Engineering Institutions.'

"(2) That the institutions in the British Isles and the Dominions named above be invited to co-operate in the Constitution of the Council.

"(3) That in the event of such invitation being issued in the name of the two Institutions, a memorandum be sent at the same time embodying a concrete proposal for the Constitution of the Empire Council for consideration. and a note explaining its suggested principal functions,

"(4) That the institutions and institutes invited to co-operate be invited as *representative units.*

"(5) That preliminary steps be taken at an early date by the Institution of Mining and Metallurgy and the Institution of Mining Engineers to convene an Empire Mining and Metallurgical Congress for 1924, to be held at the British Empire Exhibition in London.

"It is suggested that one paper be secured from each of the following countries:—Canada, Australia, South Africa, and India, written by the chief engineers of each country in collaboration, dealing comprehensively with recent developments in the mineral and metal industries of the Dominion concerned, or generally on their mining and metallurgical practice. The respective Dominions' institutes should be invited to arrange for such papers, these to be presented at the Congress in session as a whole, and remitted for discussion to the different sections."

Mr. H. F. Marriott, a member of the executive committee of the I.M.M. (Lond.), is at present making a tour of the British Dominions with the object of explaining the scheme. He has already met members of the Council of the Aus. I.M.M. and representatives of Branches of the Institute. In every case the suggestion to form an Empire Council has been approved. The matter will be mentioned on the occasion of the First Ordinary Meeting in December.

FIRST ORDINARY MEETING, 1923.

Members are reminded of the First Ordinary Meeting, 1923, which commences at Hobart, Tasmania, on 17th December. It is imperative that hotel accommodation be secured through the Secretary without delay. Passages by boat and rail should also be secured at once through the Tasmanian Government Tourist Bureau, 59 William-street, Melbourne. Particulars of hotel accommodation and of boat and rail fares were included in Circular No. 3, dated 22nd August, 1923.

NOMINATION OF PRESIDENT AND COUNCIL FOR 1924.

The following nominations for office as above have been received :—*President*—H. W. Gepp. *Councillors*—Representing Australasia: Colin Fraser, James Hebbard, A. S. Kenyon, and W. E. Wainwright; Representing New South Wales: C. C. Freeman and F. Danvers Power; Representing Victoria: H. Herman; Representing Queensland: W. H. Corbould; Representing South Australia: H. Lipson Hancock; Representing Western Australia: J. W. Sutherland; Representing Tasmania: H. W. Gepp; Representing New Zealand: D. B. Waters.

As the nominations do not exceed the vacancies to be filled, no ballot will be held.

MOVEMENTS OF MEMBERS.

A. F. S. ANDERSON, who was for some years with the Tongah Harbour Tin Dredging Co. N.L., West Siam, has severed his connection with that company, and is now at New Norfolk, Tasmania.

B. V. Barton is now in New Guinea on a visit of inspection on behalf of a Melbourne company.

Preston Chambers, of Auckland, proposes visiting Great Britain in March next, and expects to attend the World Power Conference at the Empire Exhibition in July, 1924.

J. Hopkins has resigned from the position of assistant superintendent, Experimental and Research Department, at the Broken Hill Associated Smelters, Port Pirie, and is now on the staff of the Electrolytic Zinc Co. of Australasia at Risdon-road, Hobart, Tas.

A. J. King, of the Nawng Pet Tin Co., Siam, recently paid a visit to Australia.

W. Kingston has resigned from the position of general manager of the Mount Quamby G.M. Co., North Queensland, and is now in Bairnsdale, Vic.

D. C. MacGruer is now on the staff of the Phosphate Co-operative Company of Australia, Melbourne. Prior to this appointment he was with the Sulphide Corporation Ltd. at Boolaroo, N.S.W.

D. C. Mackenzie, late Inspector of Mines, Victoria, has been appointed manager of the Riverina Collieries, Oaklands, N.S.W.

S. R. Mitchell, of Melbourne, recently returned from a tour through America.

J. R. Pound (member), lecturer in science at the School of Mines, Ballarat, has received the degree of Doctor of Science at the Melbourne University. In 1906 he won a natural philosophy exhibition, an exhibition in mixed mathematics, and an equal place in the chemistry exhibition. He graduated with honors, and won the Kernot and the Government Research scholarships.

H. R. Stafford, recently with the Morning Star Gold Mine, Wood's Point, Vic., has accepted an appointment on the staff of the Burma Corporation Ltd., Namtu, N.S.S., Burma.

Dr. F. L. Stillwell has returned to Melbourne after an extended tour of South Africa, America, and Great Britain.

BRANCH REPORTS.

BROKEN HILL BRANCH.

A meeting of the Broken Hill Branch was held on 21st August, 1923, at 8 p.m., when a paper entitled "The Equipment for a University Ore-Dressing Laboratory" was read by Mr. H. W. Gartrell of the Adelaide University. The paper included a review of the training of Mining Students.

The nomination of Mr. Maurice A. E. Mawby as a Student was approved for despatch to the Council.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

FROM 30TH JUNE, 1923, TO 30TH SEPTEMBER, 1923.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Queensland Department of Mines :		
Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines :		
Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines :		
Report of Executive	monthly	Bulawayo
Department of Mines, Victoria :		
Bulletin, No. 48, Morning Star, New Loch Fyne and Star of the West Mines, Woods Point.		
Memoirs, No. 14—The Ballarat Goldfield		Melbourne
Australian National Research Council :		
Australian Science Abstracts, Vol. II., No. 3		Sydney
Department of Mines, Queensland :		
Publication, No. 272—Geology of the Walloon-Rosewood Coalfield		Brisbane
Institution of Mining and Metallurgy :		
Bulletins, Nos. 225 and 226		London

Institution of Mining Engineers :	
Transactions, Vol. LXV., Parts 3 and 4 -	London
Geological Society :	
Quarterly Journal, No. 314 - -	London
Imperial Mineral Resources Bureau:	
Report on Graphite	
Mining Law, Vol. IV. (South Africa) -	London
Annales des Mines :	
1923, Part 6 - - - - -	Paris
Mining and Geological Institute of India :	
Transactions, Vol. XVII., Part 3 - -	Calcutta
Geological Survey of India :	
Record, Vol. LIV., Part 3 - - -	Calcutta
Canadian Institute of Mining and Metallurgy :	
Bulletin, No. 134 - - - -	Montreal
Department of Mines, Canada :	
Summary Report of Geological Survey, 1921 ;	
Bulletins, Nos. 125, 127-132	
Museum Bulletin No. 35 - -	Toronto
Department of the Interior—Bureau of Mines :	
Technical Papers, Nos. 290, 301 and 309 -	
Bulletins, Nos. 186, 187, 199 and 202 -	Washington
American Institute of Mining & Metallurgical Engineers :	
Various Papers - - - -	New York
California State Mining Bureau :	
"Mining in California" (monthly), "Summary of Operations, California Oil Wells" (monthly) -	San Francisco
United States Geological Survey :	
Water Supply Papers, Nos. 463, 473, 478-480, 482-484, 486, 480 C and D, 504, 507, 508	
Professional Papers, Nos. 122, 124, 130, 131 A,B, C,D,E,F,G,H	
Bulletins, Nos. 408, 686, 707, 722, 724, 727, 730 D, 731, 733-735 D,F,G,H,I,J, 736 C,D,E, F,G,H, 737, 742, 751 A	
Mineral Resources of the United States :	
1919, Parts I. and II., 1921	
Annual Report for year ending 30th June, 1922	

Mineral Resources (various Papers)
1920, Part I., Nos. 28, 30, Part II., Nos. 32, 34, 55; 1921, Part I., Nos. 6-8, 11., 13-30, Part II., Nos. 7-12, 18-30 ; 1922, Part II., No. 1 - New York

Colorado Scientific Society :
Proceedings, Vol. XI., pp. 1-366 - - Denver

Secretaria de Industria Y Comercio :
Bulletins, Vol. XIV., Nos. 4 and 5 - - Mexico

RECENT ARTICLES ON MINING MATTERS.

(30th June, 1923, to 30th September. 1923.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Engineering and Mining Journal Press.*

(6) *Annales des Mines*, Paris, France, mth.

(7) *Publications*, Department of Mines, Melbourne, Victoria.

(8) *Publications*, Department of Mines, Sydney, New South Wales.

(9) *Publications*, Department of Mines, Adelaide, South Australia.

(10) *Publications*, Department of Mines, Brisbane, Queensland.

(11) *Publications*, Department of Mines, Perth, Western Australia.

(12) *Publications*, Department of Mines, Hobart, Tasmania.

(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(14) *Publications*, Bureau of Mines, Toronto, Ontario.

(15) *Publications*, Geological Survey of India, Calcutta.

(16) *Publications*, Geological Survey, U.S.A., Washington.

(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(18) *Publications*, California State Mining Bureau, Sacramento, Cal.

(19) *Reports* Aust. Assoc. Adv. Science Sydney, New South Wales.

(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(21) *Quarterly Journal*, Geological Society, London.

(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.

(23) *Transactions*, Inst. Min. Eng., London.

(24) *Journal*, Canadian Mining Inst., Ottawa, Ontario.

(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(26) *Transactions*, Am. Inst. of Min. & Met. Eng., New York City.

(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(28) *Journal*, Franklin Inst., Philadelphia, Pa.

(29) *Chemical Engineering and Mining Review.*

(30) *Transactions*, Am. Soc. C.E., New York City.

(31) *Bulletins*, Société des Ingénieurs Civils, Paris.

(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(33) *Publications*, Iron and Steel Institute, London.

(34) *Proceedings*, Inst. of Mech. Eng., London.

(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(37) *Transactions*, Mining and Geological Institute of India, Calcutta.

(38) *Publications*, Department of Mines, Wellington, N.Z.

(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(40) *Publications*, Bureau of Mines, Dept. of the Interior, Washington, U.S.A.

LIST OF ARTICLES.

MECHANICAL.

Design of Furnaces for Burning Powdered Coal. (5) June 30, 1923.

Pivoted Belt Conveyor of Automobile Parts. R. F. Mundorff. (5) July 21, 1923.

Discussion of Drill Steel Papers Presented at New York Meeting, February, 1923. (26) June, 1923.

METALLURGICAL.

Sintering to Increase Returns to Ore Shippers. H. J. Stehli. (5) June 23, 1923.

Modified Method of Testing for Silver in Barren Solutions. F. C. Bond. (5) June 23, 1923.

Improvements in Zinc Milling in the Tri-State District. H. H. Wallower. (5) June 30, 1923.

An Improvement in Electrolytic Silver Refining. A. H. W. Cleave. (5) July 7, 1923.

Selective Flotation at Simon, Nev. W. Macdonald. (5) July 14, 1923.

Is Much Grinding Accomplished on the Ball-Mill Liner? H. Hardinge. (5) July 14, 1923.

Screen-Sorting Analyses by Hydrostatic Separation. S. Bagnara. (5) July 14, 1923.

Trend of Metallurgical Practice on the Witwatersrand. A. W. Allen. (5) July 14, 1923.

Soluble Silica in the Precipitation of Zinc-Sulphate Solution for Electrolysis. J. O. Bretherton. (5) July 14, 1923.

Advantages of Chile Copper Company's New Copper–Silicon Anodes. (5) July 14, 1923.

Centrifugal Force in Metallurgy. W. M'A. Johnson. (5) July 21, 1923.

Magnetic Separation of Complex Lead–Zinc Ores. J. H. Hastings. (5) July 21, 1923.

Method of Determining Zinc and Lead in Materials Containing Large Amounts of Carbon. W. Kahlbaum. (5) July 21, 1923.

Milling Manganese Ore at Algonquin Mine, Philipsburg, Montana. R. V. Ageton. (5) Aug. 4, 1923.

A Résumé of Copper Leaching in Australia. H. R. Sleeman. (5) Aug. 4, 1923.

Weight of Steel Balls. D. H. Fairchild. (5) Aug. 4, 1923.

Method of Smelting Tin Concentrate in the Far East. (5) Aug. 11, 1923.

Solving Metallurgical Problems with the Reflecting Microscope. G. M. Schwartz. (5) Aug. 11, 1923.

Fundamentals of Amalgamation. A. W. Allen. (5) Aug. 18, 1923.

Sampling Custom Ores. W. A. Linfesty. (5) Aug. 18, 1923.

Mechanical Sampling of Ore. C. D. Demond and A. C. Halferdahl. (5) July 28, 1923.

The Trail Smelter. G. J. Young. (5) July 28, 1923.

Electrolytic Copper Tank-House of the United States Metals Refining Co., Chrome, N.J. V. A. James. (24) June, 1923.

Molecular Rearrangement as a Factor in Comminution. B. W. Holman. (32) June, 1923.

Solid Solutions. W. Rosenhain. (26) June, 1923.

Proposed Plan of Crushing, Grinding, and Concentrating Low-Grade Sulphide Ore. A. Crowfoot. (26) July, 1923.

Desulphurizing Power of Iron Blast-Furnace Slags. R. S. M'Caffery and J. F. Oesterle. (26) July, 1923.

MINING.

Cost of Steam Shovel and Underground Mining of Nevada Consolidated. (5) June 30, 1923.

Crosshead for Shaft Sinking. A. A. Richie. (5) July 7, 1923.

Mining Under Fire Conditions. W. G. Scott and S. A. Mewhirter. (5) July 21, 1923.

Driving a 1,200-ft. Waste-Pass. C. F. Williams. (5) Aug. 11, 1923.

Hydraulic Stowage. T. L. M'Coll. (24) July, 1923.

MISCELLANEOUS.

Electrical Equipment of Rosario Mill and Mine. F. Krug. (5) June 30, 1923.

Marketing of Diatomaceous Earth. N. Goodwin. (5) June 30, 1923.

Park City: a Lead–Silver District in Utah. G. R. M'Kay. (5) July 7, 1923.

Speed of Moving Men in Mine Shafts. M. W. von Bernewitz. (5) July 7, 1923.

Friction Factors in the Metric System. W. S. Weeks. (5) July 7, 1923

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Gold Mining in Eastern Siberia. F. L. Cole. (5) Aug. 4, 1923.

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A Few Notes on Modern Tendencies in Coking Practice. C. E. Wallin. (24) July, 1923.

Description of the New Wash-House at No. 2 Colliery, Dominion Coal Co. Ltd., Glace Bay. W. H. Graham. (24) July, 1923.

Closed System Breathing Apparatus. J. C. Mackie. (24) July, 1923.

The Human Element in the Prevention of Accidents. J. R. Dinn. (24) July, 1923.

Papua and its People. C. G. Lanfear Thompson. (32) June, 1923.

Morning Star, New Loch Fyne, and Star of the West Mines. H. S. Whitelaw. (7) Bull. No. 48.

The Ballarat Goldfield. W. Baragwanath. (7) Memoirs, No. 14

Some Notes on the Training of Men for Rescue Teams. A. P. Veale. (23) Vol. LXV., Part 4.

The Control of Atmospheric Conditions in Hot and Deep Mines. Sir John Cadman. (23) Vol. LXV., Part 5.

PROCEEDINGS

OF THE

AUSTRALASIAN

Institute of Mining & Metallurgy

(INCORPORATED).

NEW SERIES.

No. 52.

EDITED BY THE SECRETARY.

This Institute is not responsible, as a body, for the facts and opinions advanced in any of its publications.

31st DECEMBER, 1923.

PUBLISHED QUARTERLY BY THE AUSTRALASIAN INSTITUTE OF
MINING & METALLURGY (INCORPORATED), MELBOURNE.

Australasian Institute of Mining and Metallurgy

(INCORPORATED).

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D. B. Waters, N.Z. | H. W. Gepp, Tas.

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John M. Bridge
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A. S. Kenyon.

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Secretary:

D. L. Stirling, Collins Court, 374 Little Collins Street, Melbourne, Vic.

BRANCH SECRETARIES:

Broken Hill -	C. H. J. Clayton, c/o Zinc Corporation Ltd., Broken Hill, N.S.W.
Kalgoorlie -	S. G. Turrell, 13 Boulder Road, Kalgoorlie, W.A.
Mount Morgan -	B. G. Patterson, Mount Morgan, Q.
Port Pirie -	G. Dey, c/o Broken Hill Associated Smelters, Port Pirie.
Sydney -	F. Danvers Power, 31 Hunter Street, Sydney, N.S.W.

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New Zealand -	D. B. Waters, The University, Dunedin.
Tasmania -	Russell M. Murray, Queenstown.
North Queensland	Murray Russell, Department of Mines, Brisbane.
Papua - -	W. G. Langford, c/o Anglo-Persian Oil Co., Popo, Papua.

The Executive Committee consists of all Members of the Council residing, or for the time being, in proximity to the Head Office.

Head Office:

374 Collins Court, Little Collins Street, Melbourne, Victoria.

Institute Matters.

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MINUTES OF MEETINGS

OF THE INSTITUTE.

First Ordinary Meeting, 1923, Hobart.

15TH to 22ND DECEMBER.

SATURDAY, 15TH DECEMBER.

About 70 members and students from various parts of the Commonwealth arrived at Hobart by train from Launceston on Saturday, 15th December, and were met at the Hobart railway station by members of the local committee.

SUNDAY, 16TH DECEMBER.

No special programme was arranged for Sunday, and the visitors availed themselves of the opportunity to visit the famous beauty spots and places of historic interest around Hobart, such as Port Arthur, New Norfolk, Russell Falls, the National Park, etc.

MONDAY, 17TH DECEMBER.

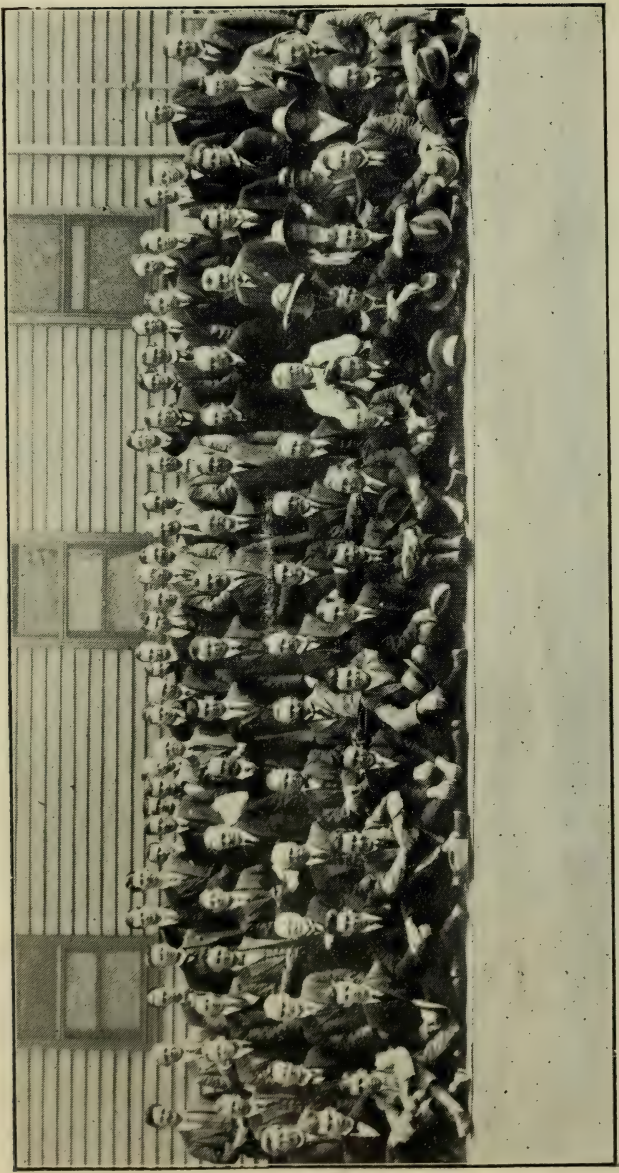
Members left Hobart by the 8.27 for an all-day inspection of the works of the Electrolytic Zinc Company of Australasia Limited at Risdon. On arrival at the works the party was met and welcomed by the general manager, Mr. H. W. Gepp. In a large marquee erected for the occasion Mr. Gepp addressed members on the growth and operations of the company. His address included a general review, largely technical, of the process and operations. The most interesting development of recent date, he said, was the decision by the board of directors to erect a superphosphate plant for the production of 50,000 tons annually of the fertilizer, which was so essential for the farming and pastoral communities. The price of the material at Risdon would be the same as in Melbourne, thereby making the price of superphosphate in Southern Tasmania very considerably lower than it had been in the past. He felt sure that, with cheaper fertilizers available, the farmers, orchardists and pastoralists would increase their consumption, and thereby increase the productivity of Tasmanian soils. The top-dressing of pasture lands was being actively carried out on the mainland with astonishingly good results—in many cases increasing the carrying capacity of the land two or threefold. They had much cause to be thankful for the success which had attended the inauguration of the industries at Risdon, thereby having beneficial effects upon all parts of Tasmania, including assurance to the West Coast of a steady development of the mineral resources of that area. Many anxious times had been experienced, and they still had much to do, but they were encouraged to persevere in the developmental work and improvements by the steady goodwill of all sections of the community in Tasmania. It was impossible to foretell the date—it might be soon—when everyone in Australia, whether he was a user of zinc, of superphosphate, or of any of the other products of the company, would have cause to be thankful that the brains and energy of Australian companies, Australian technologists, and Australian workmen had made this Commonwealth secure in her supplies of such vital products. As in the past, he wished to acknowledge the great assistance of members of their staff from the United States of America.

The party was then divided into sections of convenient size, each section being in charge of a member of the staff of the company, and proceeded to inspect the works. Luncheon was served by the company in the marquee. After the toast of "The King" had been honoured, Mr. W. E. Wainwright, member of Council, proposed a vote of thanks to the Electrolytic Zinc Company, the general manager, staff, and employés generally. Personally, he had been keenly interested in the rise and progress of the company and the worthy general manager, his friend, Mr. Gepp. Mr. Gepp was practically a Broken Hill man, and it was his Broken Hill training which had enabled him to come here. Much as they were indebted to the general manager and the employés, however, he thought they were probably even more beholden to those gentlemen who had realized the possibilities of this great industry and had been prepared to back it financially. Most people seemed to think that the success of mining depended on the work-people, and perhaps not sufficient credit was given to those who were courageous enough to find the money. His thanks were also due to the band for their most excellent music.

The vote was carried with acclamation.

Mr. G. H. Cunningham, acting deputy-general manager, in responding, said that they were privileged indeed in having such a representative body as the Institute of Mining and Metallurgy visit them to see what they had been able to accomplish in the production of zinc in the last seven years. They had had a very busy time building up their industry, and they had not done so without very sympathetic treatment from Broken Hill, Port Pirie, and the Wallaroo and Mount Lyell fertilizers. He could not help stating what they owed to the guidance of their general manager. He extended to the Institute a cordial invitation to visit Risdon again, and he anticipated that in the future they would have further subsidiary industries established, which would help to stabilize the mother industry.

Messrs. G. M'I. Hunter (chief accountant), Warlow Davies (chief engineer), and A. W. Hutchin (industrial superintendent) spoke in support of Mr. Cunningham's remarks.



First Ordinary Meeting, 1923. Members at Electrolytic Zinc Co's Works, Hobart.

During the luncheon music was provided by the Zinc Works band.

The visitors then continued their inspection of the plant and the company's village of Lutana. The opinion was expressed generally that this little village comprised one of the best housing schemes in Australasia. From conversation with the employés it was evident that the community spirit was very strong. The village included a central meeting hall, in course of erection. It is intended that this hall shall not only be used for social purposes, but will be used as a Sunday-school and church.

The party returned to Hobart by the 5.8 p.m. train.

At 8 p.m. a business meeting was held in the Masonic Hall, Murray-street. Mr. H. W. Gepp, vice-president, occupied the chair, and apologized for the absence of the president (Mr. Colin Fraser), who was on a visit to Europe.

The secretary read the notice convening the meeting.

Minutes of the Annual Meeting, 1923, were read and confirmed.

The activities of the Institute for the year were reviewed, and keen interest was shown in all of the subjects dealt with. The following is a summary of the review:—

1. *Publications.*—Papers issued by the Institute in the Proceedings comprised a number of interesting subjects on mining and metallurgy in Australasia. The authors of these papers were to be congratulated on the manner in which they had dealt with their subjects. The papers were a decided acquisition to the literature of the day, and kept members and the profession generally up to date in the progress of mining and metallurgy. The chairman impressed upon members their duty in the preparation of such papers and contribution to the discussion on papers published.

2. *Federal Engineering Council.*—This council, it was shown, consisted of representatives of the Australasian Institute of Mining and Metallurgy, the Institution of Engineers, Australia, and the Australian Chemical Institute. Its headquarters were in Sydney. The Aus. I.M.M. representatives on the Council were:—Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power, with the following "substitutes":—Messrs. Colin Fraser, H. A. Evans, and R. J. Donaldson. Three meetings of this Council were held during the year ending 30th June, 1923.

By-laws were drafted to govern procedure under the constitution of the Council.

An amendment was made to obviate the misinterpretation of certain functions of the Council, it being provided that the Council shall meet to deal with any question which may be referred to it for discussion by any of the constituent bodies or for decision by all of the constituent bodies. This amendment was unanimously approved by the councils of the constituent bodies.

Employment Bureau.—On this subject the following recommendations were adopted during the year :—

(1) That a central employment bureau, common to the three constituent bodies, be instituted.

(2) That an employment register be opened at headquarters, the secretaries of the constituent bodies supplying the general secretary of the Federal Council with the names of unemployed members, together with other requisite information.

(3) That each constituent body keep a list of its own members desiring employment.

(4) That headquarters circularize the principal employers, informing them of the institution of the employment bureau and of its scope.

(5) That, in order to meet expenses incurred, each constituent body pay a sum proportionate to its total membership, the total annual expenditure not to exceed £50.

Railway Concession Tickets.—This matter was considered at the request of the Council of the Aus. I.M.M., and representations have been made to the Railway Commissioners in each State in connection with the issue of concession tickets to delegates attending engineering conferences, etc., and a submission has been made that such concession tickets be issued to delegates attending conferences arranged by certain recognized technical societies and institutions irrespective of the number of delegates travelling from any particular State or States. At present the concession is only granted in those States from which there is a minimum number of six delegates travelling. Replies received to date indicated that the request will not be granted at present.

3. *Commonwealth Engineering Standards Association.*—The Institute has taken an active interest in this important matter and assisted in its organization. It is represented on the main committee by Messrs. W. E. Wainwright, G. H. Blakemore, and F. Danvers Power. The headquarters of the Association are in Sydney, and "organizing committees" are being established in each State.

The following extract from a letter by the acting hon. secretary of the Victorian organizing committee (Mr. Gerald Lightfoot) brings out very strongly the scope and importance of the Association :—

"As you are probably aware, steps were taken about a year ago for the creation of a Commonwealth Engineering Standards Association on lines similar to those which have been followed in other countries, especially in Great Britain, U.S. America, and Canada. The objects of the association are to take necessary steps for the preparation of standard specifications for materials, etc., used by engineers.

"The main objects of standardization are, of course, well known to you, and may be summarized thus :—

"(*a*) To cheapen manufacture.

"(*b*) To effect improvement in quality and design.

"(*c*) To increase production.

"(*d*) To reduce maintenance charges and variety of stocks.

"(*e*) To secure interchangeability of parts.

"Broadly speaking, the objects of standardization are, therefore, mass production and reduced cost per unit produced. Both producers and consumers are, therefore, vitally concerned in standardization.

"The operations of the Commonwealth Engineering Standards Association, which has its headquarters in Sydney, are directed by a main committee, which is representative of the Commonwealth and State Governments, the Institution of Engineers of Australia, the Australasian Institute of Mining and Metallurgy, the Australian Chemical Institute, and the Associated Chamber of Manufactures of Australia. The functions of that committee are purely administrative and directive, the actual detail work of preparing specifications and defining standards being done by special sectional committees, which are closely in touch with the industries and interests concerned, and which are thoroughly representative of both manufacturers and users of the particular materials, etc.

" Prior to the formation of the Standards Association the work had been initiated by the Institute of Science and Industry, which, with the assistance of the various interests concerned, prepared and published standard specifications for (*a*) structural steel sections, (*b*) railway rails and fishplates, and (*c*) tramway rails and fishplates. As soon as the Standards Association was established (under the ægis of the Commonwealth Government and working in co-operation with the Institute of Science and Industry), the work was handed over to that association. Already very considerable progress has been made, and a number of standard specifications will be issued at an early date.

" The present position in which the association stands in relation to the public, including the commercial and industrial community, demands the launching of a strong publicity campaign, which will make clear the aims and objects of the movement and the changes which it will effect in industry.

" With that object an influential ' organizing committee ' has already been established in Sydney, and steps are now being taken to form a similar committee in Melbourne. It is now proposed that the latter committee shall be constituted on the following lines :—

" *Victorian State Organizing Committee.*

Institution, &c.	No. of Representatives.
" 1. Institution of Engineers (Australia), Victorian Branch	3
" 2. Victorian Institute of Engineers	3
" 3. Victorian Chamber of Manufactures	2
" 4. Institute of Mining and Metallurgy	1
" 5. Australian Chemical Institute	1
" 6. Chamber of Automotive Industries	1
" 7. Institute of Architects	1
" 8. Melbourne Chamber of Commerce	1
" 9. Chamber of Mines, Victoria	1
" 10. Mines and Metals Association	1
" 11. Hardware Association	1
" Total	16

" The Commonwealth Government is contributing a sum of £2000 to the association during the current financial year, in addition to a special grant of £250 for the compilation of information relative to power supply. Moreover, several companies and individuals have already (without being solicited to do so) promised substantial sums towards the cost of the work.

"Support to this movement is an investment which will return, indirectly and directly, a series of benefits, increasing with the development of standardization in Australia, and with the resultant growth of industry, equivalent to, but not readily estimated in terms of hard cash. In order, however, to offer some definite return to companies, associations, individuals, etc., for contributions it has been decided that 'members' who subscribe annually not less than £2 2s. shall be entitled to—

"(*a*) Receive a copy of each standard specification.

"(*b*) Have the use of the association's 'information service' on standards, etc., in Australia and other countries.

"(*c*) Take part in the drafting of standard specifications in which they are specially interested.

"(*d*) Make proposals to the main committee for articles, materials, methods, etc., to be standardized, and for research work to be carried out to assist standardization."

The chairman and Mr. W. E. Wainwright commended this subject to members for their serious consideration, and invited their assistance in carrying out the work of the Standardization Association, which was a matter of very great importance to Australia.

4. *Empire Mining Congress, London,* 1924.—The proposal to hold a mining congress in London during the exhibition has been well received by the Council, and delegates from the Australasian Institute of Mining and Metallurgy will be appointed in February next. It is proposed that, in future, similar mining congresses be held periodically in other parts of the Empire. The benefit to the industry as a result of the deliberations of delegates from all parts of the Empire to such conventions should be very considerable. Many subjects of vital interest, not only to the industry, but to the British Empire, will thus be discussed by leading men in the mining world, representing institutes upon whose shoulders rests all the responsibility of the development of a great primary industry. During this congress it is proposed to form an Empire Council of Mining and Metallurgy.

(Extracts of a letter on this subject from G. C. Klug, dated 18th October 1923, were read. Mr. Klug was in Italy at the time of writing.)

5. *Empire Council of Mining and Metallurgy.*—Closer association between the Australasian I.M.M., the I.M.M. (London), and kindred institutions in the British Empire has always been advocated by the Council of the Aus I.M.M., and advantage has been taken of visits of various presidents to London on several occasions to further it. The opinions of the Council on these subjects have been very ably explained to the council of the I.M.M. (London) from time to time, and these opinions have always been well received. The visit of a member of the executive of the I.M.M. (Mr. H. F. Marriott) in August last shows how the idea has progressed. Mr. Marriott was the special delegate from the I.M.M. (London) and the Institution of Mining Engineers (London), and his main object was to enlist the support of the Aus. I.M.M. in the proposed formation of an Empire Council of Mining and Metallurgy. A memorandum on this subject was published in Proceedings, No. 51.

It is proposed that this Empire council should consist of two representatives of each of the institutions, and that its headquarters should be in London. This council would form a link between all of the British institutions. Each institute would retain complete control of its own affairs, as at present. It is proposed that the Empire Council be formed during the proposed mining congress in London in 1924.

6. *Paper for Mining Congress.*—The British institutes concerned in the congress have asked our Institute's co-operation in preparing composite papers from each of the British Dominions dealing comprehensively with recent developments in the mineral and metal industries, or generally on their mining and metallurgical practice. It is believed that such papers would give a real, dignified, and comprehensive start off towards the solidarity and unity of kindred British institutions. The Council of the Aus. I.M.M. has given its support to this suggestion, and has delegated the work to the publication committee. The committee is already in communication with branches and members in the matter, and the replies to date to its request for papers are encouraging. The paper to be prepared will deal with mining and metallurgical practice in Australasia.

7. *Registration of Mining Engineers.*—This question has been and is being carefully considered by the Council. Amongst mining engineers throughout the world opinions differ in regard to the wisdom of applying registration to mining engineers. Some of the reasons advanced against registration were enumerated and appeared in the last annual report of the Council. Nevertheless, it is felt that, as registration has proved beneficial to other professions, such as the medical profession, some practical scheme might be evolved. This subject may be considered by the proposed Empire Council of Mining and Metallurgy.

8. *Federal Geological Survey.*—The establishment of a Federal Geological Survey was urged by the Institute many years ago, and the proposal was again revived when the Australian National Research Council came into existence. The Prime Minister, Mr. Bruce, last year expressed his sympathy with the proposal. The National Research Council has been urged by the Institute to push this matter.

9. *Mine Managers' Certificate.* — Uniform standards of examination for mine managers' certificates and reciprocity between the various States is still being urged by the Council, and the Secretary for Mines, N.S.W., has been requested to convene a conference of Chief Inspectors of Mines for the purpose of drafting a Bill to meet the case.

10. *Reports in Prospectuses.*—Recommendations were framed by the Council last year with respect to misleading reports in prospectuses, and it was urged on the State Governments that legislation be enacted—

(1) To ensure that the prospectus of every new mining or metallurgical enterprise intended for public subscription be supported by a synopsis of the latest report (if any) by the State Geological Survey or Mines Department on the area, such synopsis to be previously approved by the Department concerned.

(2) To render it obligatory that persons making technical reports for the promotion of mining companies should state in the report their degree or other qualifications, and should state the university or other institution from which they obtained such qualifications.

(3) To urge the immediate necessity for a mining companies' law, especially relating to prospectuses, and providing that directors shall be personally liable for any wilfully misleading statements in such prospectuses.

Judging from replies received from the various Departments of Mines, it seems probable that amendments of the Companies Acts will eventuate, and the Council's suggestions met to some extent.

11. *Contributions by Companies Towards Cost of Publications.*—Recognizing the value of the papers published by the Institute to mining and metallurgical practice, several companies have agreed to contribute £10 per year towards the cost of printing. This practical assistance is very much appreciated by the Council. The companies that have so far contributed are:—Amalgamated Zinc (De Bavay's) Ltd., Broken Hill Associated Smelters, Broken Hill South Ltd., North Broken Hill Ltd., Electrolytic Zinc Co. of Australasia Ltd., Mount Lyell M. and R. Co., Pioneer Tin Mining Co., State Coal Mine (Vic.), Sulphide Corporation Ltd., Wallaroo and Moonta Mining and Smelting Co.

12. *World Power Conference*, 1924.—A communication was received through the Federal Engineering Council from the organizing director (Mr. D. N. Dunlop) of the World Power Conference, to be held in London during the British Empire Exhibition, 1924. The question of the preparation of a paper for this conference was considered, but it was thought that such work came more within the scope of the Institution of Engineers, Australia. That institute has, accordingly, with the assistance of the Commonwealth Government, undertaken the compilation from existing records of data regarding the coal, hydraulic, and oil resources of the Commonwealth and the preparation of a summary of the present position in Australia of power supply and its utilization in industry and commerce. The Australasian Institute of Mining and Metallurgy will be represented at the conference by Mr. H. W. Gepp and Mr. G. C. Klug. Briefly, the purpose of the World Power Conference is to consider how the industrial and scientific sources of power may be adjusted nationally and internationally.

13. *Increase of Members' Subscription.*—The treasurer, Mr. A. S. Kenyon, submitted a statement of the financial progress of the Institute since its inception.

THE FINANCIAL HISTORY OF THE INSTITUTE.

Year ending 31st Dec.	Revenue.	Publications.	Salaries and Rents.	Miscellaneous.	Total.	Funds.
	£	£	£	£	£	£
1893	412	—	75	166	241	171
1894	265	150	100	165	415	21
1895	72	—	—	—	—	93
1896	108	50	—	62	112	89
1897	271	180	40	109	329	31
1898	254	90	50	129	269	18
1899	264	90	72	76	238	44
1900	406	110	95	133	338	112
1901	476	162	64	130	356	232
1902	572	200	82	186	468	336
1903	448	139	89	108	336	448
1904	444	241	125	85	451	441
1905	452	239	137	88	464	429
1906	505	232	161	108	501	433
1907	485	105	183	171	459	459
1908	504	95	227	148	470	494
1909	633	179	234	139	552	575
1910	966	558	249	291	1098	443
1911	1070	733	230	267	1230	283
1912	1164	934	224	263	1421	26
1913	1374	684	286	348	1318	82
1914	1272	504	286	293	1083	271
1915	1062	285	296	225	806	527
1916	996	314	325	144	783	740
1917	862	104	304	74	482	1120
1918	887	116	300	143	559	1448
1919	937	372	300	259	931	1454
1920	1201	226	326	177	729	1926
1921	1035	381	326	411	1118	1843
1922	1076	548	323	339	1210	1709

The figures for 1923 were not yet complete; but there had been necessary increases in rent and salaries which would result in further depletion of the funds. The position might be briefly summarized thus :—

A revenue of £1,000 could be relied upon. Salaries and rent accounted for £450 ; miscellaneous expenditure, postage (a very heavy item), printing, etc., would absorb £250, while publications should amount to at least £500, thus leaving a deficiency of £200. During this year the more important mining companies had contributed £100 towards publication expenses, receiving a certain number of our Proceedings in return. A similar sum might be expected annually, reducing the deficit to £100, providing publications did not exceed £500. It was felt that it was advisable to increase the reserve funds to £2,000, and to increase the amount of printing and publication.

The financial position of the Institute was reviewed by a special meeting of the Executive Committee on 12th March last, and the following resolution adopted:—

> "That is is desirable to increase the amount of members' subscription to £3 3s. per annum; that, if the Council favours the suggestion, it take steps to submit same to the First Ordinary Meeting, 1923, before remission of the question to members for ballot in accordance with the articles of association."

A ballot by members of Council was accordingly taken, and showed that, with one exception, members of Council approved of the suggestion. The question was now submitted to members for review.

The present members' subscription was the lowest of any engineering institute of equal standing, and was the same as at the inauguration of the Institute 30 years ago, notwithstanding the fact that the cost of paper and printing alone had nearly doubled in recent years and the volume of Proceedings had increased. The Council had kept expenses down as low as possible without affecting the efficiency and status of the Institute, but found it now necessary, in order to make ends meet, to make the above recommendation.

After discussion it was resolved that the Council be recommended to submit the question of increase to members.

Branches in Tasmania.—Mr. R. M. Murray (Mount Lyell M. and R. Co., Queenstown, Tasmania) suggested that a West Coast Branch be formed, which might meet quarterly, or, say, three times a year, at different centres, and, for the fourth meeting, nominate delegates and attend the First Ordinary Meeting of the Institute. It was decided to leave the question in the hands of the local members to discuss and arrange.

Accountants as Members.—Mr. Gepp moved that Rule 12 be amended by the insertion of the words " or Mining or Metallurgical Accountant " after " Metallurgist."

After lengthy discussion the motion was carried.

First Ordinary Meeting, 1924.—Mr. H. Herman suggested that the next Ordinary Meeting be held in Victoria, with Yallourn as the principal attraction.

The suggestion was made a recommendation to the Council.

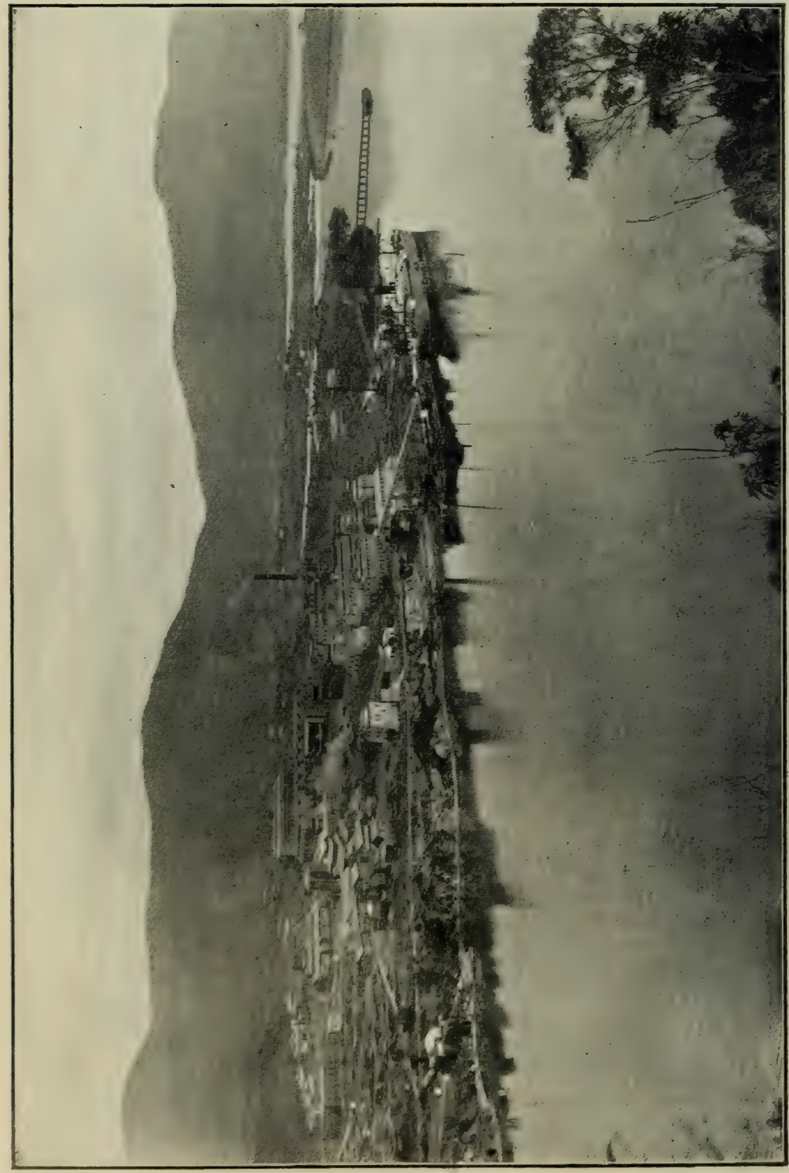
A hearty vote of thanks was carried to the Electrolytic Zinc Co., Hydro-Electric Department, Jones' (I.X.L.) Jam Factory, Cadbury-Pascall-Fry's Chocolate Works, Carbide and Electro-Products Ltd., Marine Board, the local committee of the Institute, and the honorary secretaries for their co-operation in connection with the meeting.

The meeting then closed.

TUESDAY, 18TH DECEMBER.

On Tuesday visits were made to the model factory of Cadbury-Fry-Pascall at Triffit's Point, and to the jam factory of H. Jones and Co. Ltd., in Hobart. For the trip to Cadbury's a launch was kindly placed at their disposal by the Marine Board of Tasmania. The factory is a huge, up-to-date, concrete structure situated on one of the most beautiful points on the Derwent River. The process of manufacture and the labour-saving machinery were viewed with interest. The twenty-two cottages for the work-people were a conspicuous feature. At the I.X.L. jam factory members were received by Sir Henry Jones and shown over the premises by members of the staff. The up-to-date mechanical contrivances in use were much admired. Keen interest was also taken in the workshop, where several jam-manufacturing machines were shown in the various stages of manufacture.

At 8 p.m. members met at the Masonic Hall to hear an address by the vice-president, Mr. H. W. Gepp. There was a large and representative audience, which included His Excellency the Administrator (Mr. Justice Ewing), the Premier (Hon. J. A. Lyons), Senator J. E. Ogden, Mr. D. J. O'Keefe, M.H R., Senator H. J. H. Payne, Mr. A. C. Seabrook, M.H.R., Sir Walter Lee, Colonel R. E. Snowden, M.H.A., and a number of the leading men of Hobart. Mr. W. E. Wainwright presided, and expressed pleasure at seeing such a large and important



Risdon Works of the Electrolytic Zinc Co. of Australasia Ltd.

audience. Mr. H. W. Gepp then delivered his address (which is printed in full in another part of the Proceedings), entitled "The Defence of Australia and the Relation Thereto of the Primary and Secondary Industries." At the conclusion of the address, and by the special request of Mr. Gepp, the following members contributed to the discussion thereon. The speakers included His Excellency the Administrator, Mr. W. E. Wainwright, the Premier (Hon. J. A. Lyons), Senator J. E. Ogden, Mr. D. J. O'Keefe, M.H.R., Senator H. J. H. Payne, Mr. A. C. Seabrook, M.H.R., Sir Walter Lee, Colonel R. E. Snowden, M.H.A., Messrs. J. H. Butters, O. B. Williams, and A. S. Kenyon. The proceedings terminated with a vote of thanks to the lecturer, which was carried by acclamation.

Wednesday, 19th December.

At 9.30 in the morning motor cars, carrying between 50 and 60 persons, including several ladies, left Hadley's Hotel for Fern Tree Bower, where some time was spent inspecting that beauty spot, after which the party proceeded to the Springs. On arrival there the majority of those present, excluding the ladies and the less venturesome members of the party, set out for the Pinnacle. After returning to the hotel and partaking of lunch, a move was made for the Carbide Works at Electrona, which were reached shortly after 3 p.m. On arrival the party was met by the general manager (Mr. J. H. Gillies) and staff, and subdivided into the care of guides, who conducted their charges over the plant. The electrode department was first visited, and it was explained that these essential parts of the apparatus necessary for the manufacture of carbide were first made because a supply had been impossible to obtain owing to the war. After many experiments the secret of their manufacture was solved, and now they are being turned out at the rate of four tons (eight electrodes) per shift of eight hours. The basic constituents are coke, graphite, and pitch, and with these are mixed certain other materials. First the ingredients are crushed and mixed, then heated, and placed in the press, which, by hydraulic action, exerts a pressure of 12 tons to the inch, or a cumulative pressure over the electrode of 3500 tons. After being some time in the press, the electrodes are then loaded into brick trucks and placed in a furnace, gas-heated to a temperature of 1500° C., after which they are ready for use. In the manufacture of carbide, limestone, 98 % pure, brought from 30 miles down D'Entrecasteaux Channel, is first burned in a kiln, crushed and elevated to a higher floor, where it is mixed with the required proportion of coke, and then trucked to the furnace, where a current of 25,000 amp., generating a heat of 3600° C., is used to convert the stone and coke into carbide. The molten slag is loaded into pots, where it is cooled, and then fed into a crusher, similar to an ordinary stone crusher. From this it is elevated to the bins, where it is graded and filled into tins, which, after being placed in wooden cases, are ready for shipment. The present output of the works is 1100 to 1200 cases (some 100 tons) per week, but it is intended at an early date to duplicate the plant. The works has also its own tin-making plant, capable of an output of 350 tins per day, and all necessary departments to render outside assistance unnecessary.

Carbide and Electro-products Limited, Electrona, Tasmania.

The inspection over, the party was entertained at afternoon tea by Mr. Gillies and staff, after which Mr. W. E. Wainwright, general manager of the Broken Hill South Pty. Ltd., moved a vote of thanks to Mr. Gillies and the company for their hospitality. Mr. Wainwright referred to the hard fight the Carbide and Electro-Products Company had had in arriving at a successful process. Their battle in solving the electrode manufacture was an epic feat. It had been a revelation to him to see the extent of the works and find what the industry was. Just as they were making a start the war came on, and landed the industry in trouble. Now their prosperity was at stake owing to the large supplies of carbide available from other countries, and it was necessary that adequate protection should be provided, so that the industry should be enabled to realize the big manufacturing future before it.

Mr. A. S. Kenyon predicted a great future for the industry, but it was necessary that it should be protected, as a little help at this stage would enable fine results to be attained later on. No nation could ever become great unless it had industries, and Tasmania, and also Australia, had a great asset in the carbide works. At Mr. Kenyon's invitation those present signified their appreciation of the hospitality they had received by hearty applause.

Senator H. J. M. Payne also referred to the early struggles of the company, and expressed his gratification at its progress.

Mr. Gillies, who was loudly applauded, expressed his pleasure at having the opportunity to entertain those present, many of whom had been his friends for years. He referred to the early projects of the company, the hydro-electric scheme, and the Zinc Works, and said that the former would do more in the advancement of Tasmania than anything else. The Zinc Works had made wonderful progress, and the State was lucky in having a man like Mr. Gepp at the head of affairs. Protection for the company, Mr. Gillies said, was absolutely necessary. During the war many countries manufactured carbide, and now the holders of these stocks were disposing of them at cut prices. The company had found that when it reduced prices these suppliers retaliated by reductions to rates which made profitable production by the company an impossibility. Now they were asking the Government to carry out its promise to give adequate protection. If they succeeded in weathering the storm they would be able to start other industries of which carbide was the raw material.

The party again boarded their cars, and, after a pleasant run through Margate and Kingston, reached Hobart about 6.30 p.m.

An enjoyable evening was spent at the Masonic Hall, when the visiting members and a number of representative citizens were entertained by the Electrolytic Zinc Company at a smoke social. The evening was spent in the enjoyment of music, supper, and good stories. Apart from the set programme, the Zinc Works orchestra, under the able conductorship of Mr R. S. M'Arthur (electrical engineer, Zinc Works), filled in other moments with excerpts from their extensive repertoire.

THURSDAY, 20TH DECEMBER.

Members assembled at the Masonic Hall at 10 a.m. for the purpose of reading and discussing papers prepared for the meeting. As, however, the majority of members expressed a desire to visit places of interest in and around Hobart, the chairman (Mr. H. W. Gepp) suggested that the papers be taken as read. The suggestion was adopted, and members were asked to peruse the papers carefully on their return home and forward to the secretary written contributions to the discussion. The meeting then adjourned, and members went their various ways—some to continue inspection of the Electrolytic Zinc Works, and some to visit the beauty spots of the district. At the invitation of Mr. Chester Lord, manager for Tasmania of the Hume Pipe Co., a visit was paid to that company's factory at Moonah. Motor cars provided by Mr. Lord conveyed the party to the factory, where an inspection was made of the following:—Sandstone and screenings, oiling cases and casting reinforcements, mixing aggregates, moulding pipes, making reinforcements, steam chambers, and manufactured articles. On return the visitors were the guests of the company at afternoon tea at Hadley's Hotel.

At the invitation of His Excellency the Administrator (Mr. Justice Ewing) a party of about 30 visited the State Government House, where they were received by Her Excellency (Mrs. Ewing) and entertained at morning tea. The visitors were subsequently shown over the house and grounds.

At 8 p.m. the Institute dinner was celebrated at the Carlton Club Hotel. The gathering was a large and distinguished one, representing as it did the mining, commercial, and political circles of Tasmania and the mainland. Those present included His Excellency the Administrator (Mr. Justice Ewing), the Premier (Hon. J. A. Lyons), the Chief Secretary and Minister for Mines (Hon. J. A. Guy), Mr. A. Lawson (Honorary Minister), the Master Warden (Mr. T. Murdoch, M.L.C.), Senator Ogden, the Hon. Jas. Belton, M.H.A., Sir Walter Lee, Sir Elliott Lewis, and the Mayor (Alderman J. Soundy).

Mr. H. W. Gepp, vice-president of the Institute, presided. He read a cable from Mr. Colin Fraser (president), who was at the time in Europe, sending his greetings, and submitted numerous apologies and expressions of goodwill from members of the Institute on the mainland and New Zealand. After an excellent dinner the toast of "The King" was honoured.

The Premier, in proposing the toast of "The Mining Industry and the Australasian Institute of Mining and Metallurgy," extended a cordial welcome to Hobart, on behalf of the Government and the people, to the visitors. It was appropriate, he said, that Tasmania should be chosen for a visit from the leading mining and metallurgical men of Australia, for there were few countries that had such vast and varied mineral resources. He thought that the individual spirit was passing out, and that the spirit of co-operation was coming. Mr. Gepp's address earlier in the week was an illustration of the fact that members of the Institute recognized that their interests were not merely in the industries in which they were engaged, but in Australia as a nation. He hoped that the spirit would grow and permeate Australia. So far as the mining industry was concerned, it meant a great deal to Tasmania. He wished that some important mining discovery would be made such as those that in the past had helped the State through gloomy periods. The State was sound enough now, but unfortunately the Treasury was all wrong. Sir Walter Lee could bear him out in that. He referred to the fact that the president-elect of the Institute (Mr. H. W. Gepp) was going to England shortly, and said he hoped that when he did he would tell the people there of the wonderful possibilities of Tasmania from a mining point of view.

The Minister of Mines, in responding to the toast, said he believed that the mining industry was one of the factors which was capable of making Tasmania. In past years the worth of mineral production of the State had reached £1,500,000 a year, but in the last two years it had decreased to about £800,000 a year. Providence had been good to the country in giving it great mineral wealth. It remained for them to develop the resources. Many deposits were yet undiscovered, and they were confident that Tasmania would some day be a veritable beehive of industry, largely as a result of mining.

Mr. W. E. Wainwright, general manager of Broken Hill South Ltd., responding on behalf of the Institute, referred to the cordial reception received by the visiting members. He went on to trace the history of the Institute, and said that it now had a membership of 550, including about 70 students. The chief advantage of the body was that of education, and they were now paying considerable attention to mining hygiene. realizing that it meant longer life for the employés and less compensation. The Institute also sought to protect the public in the matter of prospectuses issued in relation to new ventures. The mention of this suggested the present search for liquid oil, and in this connection he hoped that the search in Tasmania was founded on better premises than in the case of the mainland. Engineering standardization was another matter in which they were interesting themselves, and he hoped that the gentlemen present would do all in their power to assist this movement. He had to congratulate the people of Tasmania, and particularly those who had put money into the Electrolytic Zinc Company, in their judgment in selecting Mr. Gepp as manager. He had made an undoubted success of the electrolytic zinc industry. Then there was Mr. J. H. Gillies, to whom they were indebted for the germ of the hydro-electric idea and the complex ores industry. Members had been greatly impressed with what Mr. Gillies had accomplished in the establishment of the carbide industry, the path of which had been strewn with obstacles. The industry was a valuable one, and should survive its present trouble and prosper.

Mr. H. Herman (engineer in charge of briquetting and research, Victorian Electricity Commission) proposed the toast of "The Federal and State Parliaments." He said that if there was any rivalry between the Victorian Electricity Commission and the Tasmanian Hydro-Electric Department it was very friendly rivalry, such as existed between two batsmen on the cricket field. Each State should be independent in the matter of power supplies to run its industries. Tasmania and Victoria now had reliable supplies, and had eliminated three-quarters of the causes which hitherto had caused stoppages. He had received a message from Sir John Monash, who was in charge of the Victorian scheme, sending his good wishes, and expressing the hope that the Tasmanian Hydro-Electric Department and the industries dependent upon it would prosper and grow. Later, Sir John Monash intended to come to Tasmania and see the scheme for himself. Reverting to the toast, he advocated the passing of constructive criticism on Parliaments.

Senator J. E. Ogden, responding on behalf of the Federal Parliament, quoted instances in which mining had provided the magic touch that had brought prosperity and progress to barren places. He mentioned how, when the Complex Ores Bill came before the House of Assembly, a Northern member moved that in the event of the undertaking coming to fruition its activities should not be brought within 10 miles of Launceston. Now the people were chasing everywhere for its advantages. People scarcely realized the burden which the State was bearing as a result of having expended £3,000,000 on its hydro-electric scheme. He went on to say that, in his opinion, the struggle for existence was going to become keener with the passing of the years, and unless they set themselves more seriously to the problems of life it would go hard with the nation. Luxury and ease had caused the downfall of great nations, and he often wondered if history would repeat itself in their case. He hoped not. He sincerely hoped that a new public spirit would be born in Australia. The people should realize that they had a country to develop and a great nation to make, and that it was a public duty to give service to the nation.

Mr. Gepp said he was sure that if the majority of their representatives in Parliament had the same spirit and intentions as Senator Ogden Australia would be all right.

Sir Walter Lee, also responding, said that politicians as a rule did their best to carry out the duties entrusted to them by their constituents. He did not think that improvement would be brought about by a system of electoral Ministries. He agreed with Senator Ogden, however, to the extent that the people ought to take a greater interest in the public affairs of the State. The great trouble was the apathy of the people. He would like to see the party principle strengthened. Party organization also should be strengthened, and a greater amount of discipline should be exercised over members of Parliament than at present. People should make up their minds that they were going to return to the Parliament of the State and Commonwealth men of strong conviction, like Senator Ogden—men who stood for right and clean thinking, no matter what their party beliefs might be. In that case the Parliaments of the State and Commonwealth would be cleaner and stronger, and the people would get better legislation and administration than in the past. If that could be accomplished, he was confident that there was a bright future for this State.

The toast of "Kindred Institutions" was proposed by Mr. R. M. Murray, general manager Mount Lyell Mining and Railway Co., the toast being coupled with the names of Mr. Bertram Wood (vice-chairman of the Tasmanian division of the Institution of Engineers, Australia), Mr. J. R. Johnston (president of the Chamber of Commerce), and Mr. F. Lindsay Gunn (president of the Chamber of Manufactures).

Mr. Murray spoke of the industries which had been established in the State, the Electrolytic Zinc Works occupying a very important position, and, in mentioning industries engaged in on the mainland, expressed the hope that the whole of the output at Risdon would be used in Australia. The Chamber of Manufactures represented all the various industries, and, generally speaking, manufacturing industries were on a better wicket than primary industries, one reason being that manufactures had the advantage of protective duties. There should, however, be concerted effort on the part of primary and secondary industries. Manufacturers must manufacture as cheaply as they could and give the resultant advantage to primary producers, so that, in turn, primary producers might produce as cheaply as possible.

Mr. Bertram Wood, in responding, eulogized the industrial and engineering achievements of Tasmania, and remarked that one of the finest examples of concrete work in the world would be seen by those who were to visit the Great Lakes on the morrow. The crossing of the Derwent at Risdon by the electric cables was also a fine piece of engineering work, and further creditable examples were found in the beautiful scenic railway up the King River and the Lake Margaret hydro-electric plant.

Mr. J. R. Johnston said that the Chamber of Commerce claimed to take an interest in mining and metallurgical institutes, and that it was always prepared to lend its assistance to any institution or enterprise which was calculated to promote the interests of the State. In that connection he did not think they could over-estimate the value of the hydro-electric undertaking. They were proud of their electric power, and also of the great zinc works at Risdon, which, Mr. Gepp told them, were only in their infancy, and if the zinc works and other undertakings progressed as was anticipated there would be no holding Tasmania back. They regarded Mr. Butters, the general manager of the Hydro-Electric Department, and Mr. Gepp, general manager of the Zinc Works, as their supermen. One was a man of metal, and the other a man of power, and in conjunction with the efforts of other super-industrialists throughout the State there was little doubt that they would keep Tasmania to the fore.

Mr. F. Lindsay Gunn directed attention to the need for developing their secondary industries, and said that that was where the Chamber of Manufactures came in. He emphasized the necessity for increasing the population of the State, and contended that the development of secondary industries afforded the best opportunity for settling migrants. To his mind, too much money had been spent on placing people on the land, to the detriment of secondary industrial pursuits An immense amount of money was spent in importing manufactures from overseas, and much of the imported machinery and other manufactured goods might be made in Australia, to the great benefit of the Commonwealth.

The toast of "The City and Port of Hobart" was proposed by Mr. A. S. Kenyon, who pointed out that, valuable as the Hydro-Electric Department was in pushing the State ahead, the city and port could with advantage come in and assist in the establishment of means of transit and associated matters.

The Mayor (Alderman J. Soundy), responding for the city, alluded to the remarkable growth of Hobart during the past few years. When they spoke of Tasmania's financial difficulties, he said, they did so with bated breath, but difficulties and problems were not unmixed evils. It was the facing of difficulties, and the solving of problems, which developed strength of character and purpose, and it was such difficulties and problems which created the right men to deal with them, that would mean the success not only of Hobart but of the State.

The Master Warden (Hon. T. Murdoch, M.L.C.) responded for the port, and in drawing attention to the endeavours made by the Marine Board to secure the establishment of industries in the district, remarked that no charge was made for machinery brought into the port for industrial construction purposes. That was the policy the Board was willing to continue with, and it was prepared to offer the same inducement to any industries which might be offering. The Board assisted the Electrolytic Zinc Company in granting it the use of the water-front at Risdon for its works without charge. One other point he would like to mention. The "Australian Year Book" gave the value of the Tasmanian imports and exports at 3½ millions sterling, but, according to the State Statistician (Major Giblin), the imports and exports came to more than 17½ millions sterling; the balance of trade represented some 13 millions sterling going to Victoria and New South Wales, the goods concerned being transhipped or used there.

"The Press" was toasted at the call of the Chairman, and was responded to by the associate editor of the *Mercury* (Mr. L. Broinowski) and Mr. P. G. Tait.

Mr. D. H. Brown (Melbourne University) returned thanks on behalf of students from the mainland States and of Tasmania for the opportunity afforded them of visiting various industrial works and factories.

Mr. J. H. Butters proposed the toast of "The President of the Australasian Institute of Mining and Metallurgy," and coupled the names of Mr. Colin Fraser (general manager of the Broken Hill Associated Smelters Ltd.) and Mr. Gepp.

Mr. Gepp informally replied, and this concluded the proceedings.

FRIDAY, 21ST DECEMBER.

The whole of Friday was occupied by a visit to the Hydro-Electric Works (which are owned by the Government of Tasmania) at Waddamana and the Great Lake. The party numbered about 70, and travelled by motor car. Half of the party went as far as the Lake—a distance of about 100 miles from Hobart; the other half went direct to Waddamana—a distance of about 75 miles from Hobart. Both parties joined forces later at Waddamana power station, where they were shown over the works by Messrs. Curtis, Halkyard, and Evans. (A paper by Mr. C. C. Halkyard, entitled "A Few Notes on the Hydro-Electric Power of Tasmania," is published in this issue of the Proceedings. This paper gives, in a concise manner, the history of the scheme, and describes the works in detail).

Mr. J. H. Butters, general manager of the Hydro-Electric Department, personally conducted the leading members of the party, and entertained the visitors at luncheon at the accommodation house at Waddamana. After a very enjoyable trip the party returned to Hobart—some at 8 p.m. and others at 11 p.m.

This concluded one of the most important meetings in the history of the Institute. The success of the meeting was largely due to the hospitality extended to members of the Institute by the industrial companies and the leading citizens of Hobart, and to the admirable arrangements made by the local committee.

OF THE COUNCIL

At Masonic Hall, Hobart, Tas.

DECEMBER 17TH, 1923—10 a.m.

Present:—Messrs. H. W. Gepp (in the chair), W. E. Wainwright, H. Herman, A. S. Kenyon, R. M. Murray. (Substitutes—O. B. Williams, H. H. Schlapp, W. E. Hitchcock, H. Warlow Davies, and C. G. Hylton.)

The following subjects were submitted for consideration:—

(1) Papers published.
(2) Federal Engineering Council.
(3) Engineering Standardization.
(4) Empire Mining Congress and Empire Council of Mining and Metallurgy.
(5) Federal Geological Survey.
(6) Mining Managers' Certificates.
(7) Reports in Prospectuses.
(8) Formation of Branches in Tasmania.

On account of the very full discussion on these subjects at the preceding meeting of members, in which members of the Council joined, it was agreed that the recommendations made at that meeting go forward for consideration by the Executive Committee during the year.

The principal recommendations were in regard to—

(*a*) Proposed increase of the amount of members' subscriptions—viz., "That the Council submit to all corporate members for ballot that their subscription be increased by £1 1s."

(*b*) Accountants as Members.—"That Rule 12 be amended by the insertion of the words, 'or mining or metallurgical accountant' after 'Metallurgist.'"

(*c*) First Ordinary Meeting, 1924.—"That the First Ordinary Meeting, 1924, be held in Victoria, with Yallourn as the principal attraction."

(*d*) Branches in Tasmania.—"That the establishment of Branches in Tasmania be left in the hands of local members to discuss and arrange."

OF THE EXECUTIVE COMMITTEE.

(Summary.)

OCTOBER 29TH, 1923—1 P.M.

Minutes of the previous meeting were confirmed.

The secretary's report was presented, and accounts to the amount of £188, including petty cash, £13, salary and rent (two months), £66 13s. 4d., Broken Hill Branch, £25, reporting conference, £9 12s., bookcases, £6, printing Proceedings and sundries, £67, were passed for payment.

As a result of ballot by members of Council, the following were admitted to the Institute :—As Associate Member—Mr. John Hopkins. As Juniors—Messrs. Alan Trevor Davies and John Arthur Blackett Forster.

Nominations as follow were approved :—As Members—Messrs. Augustus Frederick Heseltine and Richard Arthur Jenkin. As Junior—Mr. Michael Francis Larkin.

Application by Mr. W. G. A. Smith for transfer from Associate Member to Member was again considered. The application was refused.

The following papers were submitted and referred to the Publication Committee :—" The Hydrogen Roasting of Pyritic Ores, with Recovery of Sulphur and Iron," by Hartwell Conder ; " Comparative Cost of Rock Drilling," by Thos. G. Hanton ; " The Action of Light on Cinnabar," by W. H. Cropp.

A donation of books to the library by Mr. A. T. Brown, of Melbourne, was reported, and a vote of thanks to Mr. Brown was recorded. The gift included a complete set of Transactions of the American Institute of Mining and Metallurgy.

Report of progress of the First Ordinary Meeting, 1923, was submitted.

Directions were given in regard to a meeting of the full Council to be held at Hobart at the time of the First Ordinary Meeting. It was decided that an agenda for the full Council meeting be prepared and issued at the end of November.

A letter was submitted containing a proposal for the housing of all technical societies in Melbourne in one building. The suggestion was favourably entertained, and it was decided to join with representatives of other institutes when a conference was called to discuss the question.

Co-operation was invited by the Institution of Municipal Engineers in the matter of securing counsel's opinion in regard to the unsatisfactory replies received by certain engineers to their application for registration under the *Architects' Registration Act.* Resolved, that fuller information be obtained, particularly in respect to probable effect on members of the Aus. I.M.M.

Mr. H. W. Gepp was appointed (in place of Mr. C. F. Courtney) to act with Mr. G. C. Klug as the Institute's representative at the World Power Congress to be held in London in 1924. Resolved, that the press be notified of the Institute's representation at this congress.

A letter was submitted from Mr. G. C. Klug (London) reporting progress in regard to the proposed Register of Mining Engineers and Metallurgists, under the direction of the proposed Empire Council of Mining and Metallurgy. Resolved, that Mr. Klug be thanked for his communication and supplied with a full report of the interview with Mr. H. F. Marriott on 16th August.

It was decided that a special meeting of the Executive Committee be convened to consider and decide as to the scope and preparation of a composite paper for presentation to the Empire Mining and Metallurgical Congress to be held at the British Empire Exhibition, in London, in 1924.

NOVEMBER 26TH, 1923—1 P.M.

Minutes of previous meeting were read and confirmed.

The secretary's report was presented, and accounts to the amount of £69, including petty cash, £6; advance expenses First Ordinary Meeting, 1923, £50; insurance, £1 12s. 6d.; and printing, £10.

As a result of ballot by members of Council, the following were declared elected to the Institute:—As Members—Messrs. Augustus Frederick Heseltine and Richard Arthur Jenkin. As Junior—Mr. Michael Francis Larkin.

Nominations as follow were approved for ballot:—As Members—Messrs. John Archibald Campbell Bayne and George Hamilton Cunningham. Transfer, Associate Member to Member—Mr. Walter Gilbert Langford. As Associate Member—Mr. Bernard Cedric Job.

The following papers were submitted and referred to the Publication Committee:—"Rock-Drill Steel Problem in Australia, with Some Special Steps to Its Solution," by G. L. Thirkell; "Dorr Appliances," by P. R. Middleton; "An Outline of the Economic Geology of Tasmania," by Loftus Hills.

A progress report in regard to the preparation of a composite paper for the Mining Congress in London in 1924 was submitted, and it was resolved that each member of Council be supplied with a copy of the report, also each Branch and member already invited to prepare articles for the paper.

Progress was reported in respect to the First Ordinary Meeting, 1923.

Resolved, that the agenda of the meeting of the full Council, to be held at Hobart on 17th December, include a review of the following subjects:—(1) Papers, (2) Federal Engineering Council, (3) Engineering Standardization, (4) Empire Mining Congress and Empire Council of Mining and Metallurgy, (5) Federal Geological Survey, (6) Mining Managers' Certificates, (7) Reports in Prospectuses, (8) Formation of Branches in Tasmania, (9) Expression of Appreciation of assistance in connection with the Hobart Meeting, (10) Any other business that may be submitted.

Arrangements were made for the appointment of substitute Councillors to attend the meeting of the full Council on 17th December.

Papers for the Students' prize, 1923, were submitted, and the following were appointed examiners:—Messrs. H. W. Gepp, W. E. Wainwright, and J. M. Bridge.

Resolved, that the annual meeting of members be held on Monday, 31st March, 1924.

NOTICES.

The rooms of the Institute are open from 9 A.M. to 5 P.M. daily, except Sundays and Public Holidays.

MEMBERSHIP.

The following is a list of members admitted to the Institute since the issue of Proceedings, No. 51 :—

Members.

Names and Addresses.

HESELTINE, AUGUSTUS FREDERICK, School of Mines, Ballarat, Vic.

JENKIN, RICHARD ARTHUR, Mount Bischoff Smelting Works, Launceston, Tas.

Associate Member.

HOPKINS, JOHN, c/o Electrolytic Zinc Co. of Australasia Ltd., Risdon-road, Hobart, Tas.

Juniors.

DAVIES, ALAN TREVOR, " Warragul," Main-road, Newtown, Hobart, Tas.

FORSTER, JOHN ARTHUR BLACKETT, Pressland House, Hobart, Tas.

LARKIN, MICHAEL FRANCIS, c/o Magnet Silver Mining Co., Magnet, Tas.

EMPIRE MINING AND METALLURGICAL CONGRESS, LONDON, 1924.

Viscount Long of Wraxall has accepted the presidency of the forthcoming Empire Mining and Metallurgical Congress to be held at the British Empire Exhibition on 3rd to 6th June (inclusive), 1924, of which the Prince of Wales is honorary president. The following official personages have accepted invitations to become honorary vice-presidents of the congress :—The Secretary of State for the Colonies, the Secretary of State for India, the Secretary for Mines, the Prime Ministers of Canada, Australia, New Zealand, and Newfoundland, the High Commissioners of the Dominions and British India, and the Lord Mayor of London. The presidents of the seven convening bodies will act as vice-presidents, and will preside over the sections with which they are concerned. The Aust. I.M.M. will be officially represented at the congress by Mr. H. W. Gepp (president-elect) and Mr. G. C. Klug (member of Council). Composite papers on mining and metallurgical subjects will be submitted from each of the Dominions, and the Council of the Aus. I.M.M. is arranging for the preparation of such a paper from Australasia.

EMPIRE COUNCIL OF MINING AND METALLURGY.

Mr. H. W. Gepp (president-elect) and Mr. G. C. Klug (member of Council) have been nominated to represent the Institute on the Empire Council of Mining and Metallurgy which it is proposed to form in London during the Empire Congress of Mining and Metallurgy.

WORLD POWER CONFERENCE, 1924.

Messrs. H. W. Gepp and G. C. Klug will represent the Aus. I.M.M. at this conference, which will be held in London during the Empire Exhibition, 1924. Mr. Gepp will also represent the Commonwealth Government.

INCREASE OF MEMBERS' SUBSCRIPTIONS.

A discussion on the proposal to increase members' subscriptions took place at the First Ordinary Meeting, 1923, at Hobart, and is reported in the minutes of that meeting. The meeting recommended—

> "That the Council submit to all corporate members that their subscription be increased by one guinea."

BRANCHES IN TASMANIA.

The question of formation of Branches in Tasmania was discussed by members at the Hobart meeting. The suggestion was approved, and it was decided to leave the question in the hands of local members to discuss and arrange.

FIRST ORDINARY MEETING, 1924.

Members at the Hobart meeting recommended—

> That the First Ordinary Meeting, 1924, be held in Victoria, with Yallourn as the principal attraction.

ACCOUNTANTS AS MEMBERS.

A discussion on this matter is reported in the minutes of the First Ordinary Meeting, 1923. At this meeting it was recommended—

> That Article 12 be amended by the insertion of the words "or mining or metallurgical accountants" after the word "metallurgist."

Should this recommendation be adopted, accountants of important mining or metallurgical corporations would be eligible for admission to the Institute.

DONATION OF BOOKS.

Mr. A. T. Brown, of Melbourne, has presented to the Institute library a complete set of Transactions of the American Institute of Mining and Metallurgical Engineers, a number of volumes of "The Mineral Industry," and several other valuable publications.

COMMONWEALTH ENGINEERING STANDARDS ASSOCIATION.

Dr. H. Herman (member of Council) has been nominated to represent the Aus. I.M.M. on the Victorian State Organizing Committee of the Commonwealth Engineering Standards Association.

ANNUAL MEETING, 1924.

As already announced, the Annual Meeting, 1924, will be held at 374 Little Collins-street, Melbourne, on 31st March, at 1 p.m. The business of the meeting will be—(1) to receive and consider the report of the Council and statement of accounts for 1923; (2) to appoint auditor for 1924.

ELECTION OF PRESIDENT AND COUNCIL, 1924.

Mr. H. W. Gepp has been elected President of the Institute for 1924, and the following have been elected to fill other annual vacancies on the Council:—Representing Australasia: Messrs. Colin Fraser, James Hebbard, A. S. Kenyon and W. E. Wainwright. Representing New South Wales: Messrs. F. Danvers Power and C. C. Freeman. Representing Victoria: Dr. H. Herman. Representing Queensland: Mr. W. H. Corbould. Representing South Australia: Mr. H. Lipson Hancock. Representing Western Australia: Mr. J. W. Sutherland. Representing Tasmania: Mr. H. W. Gepp. Representing New Zealand: Professor D. B. Waters.

MOVEMENTS OF MEMBERS.

GORDON LINDESAY CLARK has been appointed lecturer in mining at the Melbourne University.

J. D. Connor, for some time Director of the Technical School, Wonthaggi, Vic., is now at the Technical School, Footscray.

Victor T. Edquist is at present on the staff of the Sons of Gwalia Ltd., Gwalia, W.A.

Chas. C. Freeman, for some years assistant superintendent of the Zinc Corporation Ltd., Broken Hill, N.S.W., has been appointed general superintendent of the Società Anonima Miniera, Cave del Predil (Raibl), Fruili, Italy.

H. W. Gepp, general manager, Electrolytic Zinc Co. of Australasia Ltd., is leaving on a visit to London in March. He has been nominated as one of the Institute's representatives on the Empire Mining and Metallurgical Congress.

H. Herman, B.C.E., M.M.E., engineer in charge of briquetting and research, State Electricity Commission, Victoria, has received the degree of Doctor of Science of the Melbourne University. Amongst the papers submitted in support of his candidature were—" The Structure of the Bendigo Goldfield," " Australian Tin Lodes and Tin Mills," and " Brown Coals of Victoria." Dr. Herman was for some years Director of the Geological Survey of Victoria.

Loftus Hills has been honoured with the degree of Doctor of Science of the University of Tasmania. The thesis submitted for the degree was entitled " The Metallogenic Epochs of Tasmania." Dr. Hills was formerly Government Geologist of Tasmania, and is at present residing in Melbourne.

W. Kingston, at one time Director of the School of Mines, Bairnsdale, is managing the Mount Jasper mine, Heazlewood, Tas.

D. S. Lawrance is now on the staff of the Broken Hill Proprietary at the iron and steel works, Newcastle, N.S.W.

W. H. Mawdsley, late of Rockhampton and Adelaide, is at present on a visit to England.

BRANCH REPORTS.

BROKEN HILL BRANCH.

An Ordinary Meeting of the Broken Hill Branch was held on Tuesday, 27th November, 1923, at 8 p.m.

Mr. F. Voss Smith presided.

Correspondence from the secretary was read and received.

A letter from Mr. G. C. Klug (London) in regard to registration was read.

As suggested by the Council, it was agreed that the Branch prepare a paper on mining and metallurgy at Broken Hill, to form part of the composite paper by members of the Institute for the Empire Mining and Metallurgical Congress, 1924. Messrs. C. H. J. Clayton and W. B. Chomley were deputed to prepare the mining and metallurgical sections of the paper respectively.

Consideration was given to the question of a nomination to fill the casual vacancy on the council caused by the retirement of Mr. C. C. Freeman. It was agreed that, with Mr. A. Molin's consent, he be nominated for the position; failing him, Mr. C. H. J. Clayton be recommended for appointment.

Mr. C. H. J. Clayton read a few interesting notes on "Rock-Drill Sharpening," comparing the coke furnace with the open coal forge.

The hon. secretary (Mr. C. H. J. Clayton) submitted his resignation and asked for a nomination of Branch secretary.

PORT PIRIE BRANCH.

At a meeting of the Port Pirie Branch, held in September last, the following officers were appointed:—

Chairman: Mr. H. St. J. Somerset.
Vice-Chairman: Mr. R. Beckwith.
Committee: Messrs. O. H. Woodward, A. J. Hemingway, P. J. Walsh, T. D'A. Melhuish, and R. V. Berriman.
Secretary: G. Dey.

The committee was instructed to draw up a set of rules to govern the proceedings of the Branch.

LIST OF PUBLICATIONS ADDED TO THE LIBRARY

From 30th September, 1923, to 31st December, 1923.

Chemical Engineering and Mining Review	monthly	Melbourne
Industrial Australian & Mining Standard	weekly	Melbourne
Engineering and Mining Journal Press	weekly	New York
Mining Magazine	monthly	London
Indian Engineering	weekly	Calcutta
Engineering Production	weekly	London
Chemical News	weekly	London
South African Engineering	monthly	London
Journal of Industrial and Engineering Chemistry	monthly	Easton, Pa.
Society of Chemical Industry : Journal	monthly	London
Chemical, Metallurgical and Mining Society of South Africa : Journal	monthly	Johannesburg
Franklin Institute : Journal	bi-monthly	Philadelphia
Institution of Mechanical Engineers : Journal	monthly	London
Queensland Department of Mines :		
Government Mining Journal	monthly	Brisbane
Transvaal Chamber of Mines :		
Monthly Analysis of Gold Production		Johannesburg
Rhodesia Chamber of Mines :		
Report of Executive	monthly	Bulawayo
Australian National Research Council :		
Australian Science Abstracts, Vol. II., No. 4		Sydney
Department of Mines, Queensland :		
Annual Report of Under Secretary for 1922		
Geological Survey Publication, No. 274		Brisbane
Department of Mines, South Australia :		
Mining Review, No. 38		Adelaide
New Zealand Board of Science and Art :		
The N.Z. Journal of Science and Technology, Vol. VI., Nos. 2 and 3		Wellington
Institution of Mining and Metallurgy :		
Bulletins, Nos. 227-230		London

Institution of Mining Engineers :	
Transactions, Vol. LXV., Parts 5 and 6, Vol. LXVI., Parts 1 and 2 - - -	London
Geological Society :	
Quarterly Journal, No. 315 - -	London
Geologists' Association :	
Proceedings, Vol. XXXIV., Part 3 - -	London
Institution of Chemistry of Great Britain and Ireland :	
Journal and Proceedings, August, 1923 -	London
Iron and Steel Institute :	
Journal, Vol. CVII., No. 1 - -	London
Royal Dublin Society :	
Scientific Proceedings, Vol. XVII., Nos. 11-31	Dublin
Societa Geologica Italiana :	
Bulletin, Vol. XLI., Part 3 - - -	Rome
Annales des Mines :	
1923, Parts 7-10 - - - -	Paris
Geological Survey of India :	
Records, Vol. LV., Parts 1 and 3 - -	Calcutta
Canadian Institute of Mining and Metallurgy :	
Monthly Bulletin, Nos. 135-138 - -	Montreal
Nova Scotian Institute of Science :	
Proceedings and Transactions, Vol. XV., Part 2	Halifax
American Institute of Mining & Metallurgical Engineers :	
Papers, Nos. 1251-1279 - - -	New York
United States Geological Survey :	
Water Supply Papers, Nos. 469, 495, 496 Bulletins, Nos. 717, 718, 738, 741, 743, 745, 750A Mineral Resources, 1922, Part 1, No. 1 ; Part 2, No. 2 - - - - -	New York
California State Mining Bureau :	
"Mining in California" (monthly) "Summary of Operations, California Oil Wells" (monthly) - - - -	San Francisco
Secretaria de Industria Y Comercio :	
Bulletins, Vol. XIV., No. 6 - -	Mexico

RECENT ARTICLES ON MINING MATTERS.

(30th September, 1923, to 31st December. 1923.)

NOTE.—*This list is prepared for the purpose of placing before members the titles of the more important papers appearing in the usual publications concerned with mining engineering, metallurgy, &c., due regard being had to Australasian requirements.*

LIST OF PUBLICATIONS.

References are given by the number prefixed to each publication in the attached list. Wk., weekly. mth., monthly.

(1) *The Industrial Australian & Mining Standard*, Melbourne, Victoria, wk., 6d.

(2) *The Queensland Government Mining Journal*, Brisbane, mth., 6d.

(3) *Metallurgical and Chemical Engineering*, New York, mth., 25c.

(4) *The Mining Journal*, London, E.C., wk., 6d.

(5) *Engineering and Mining Journal Press.*

(6) *Annales des Mines*, Paris, France, mth.

(7) *Publications*, Department of Mines, Melbourne, Victoria.

(8) *Publications*, Department of Mines, Sydney, New South Wales.

(9) *Publications*, Department of Mines, Adelaide, South Australia.

(10) *Publications*, Department of Mines, Brisbane, Queensland.

(11) *Publications*, Department of Mines, Perth, Western Australia.

(12) *Publications*, Department of Mines, Hobart, Tasmania.

(13) *Publications*, Geological Survey, Canada, Ottawa, Ontario.

(14) *Publications*, Bureau of Mines, Toronto, Ontario.

(15) *Publications*, Geological Survey of India, Calcutta.

(16) *Publications*, Geological Survey, U.S.A., Washington.

(17) *Publications*, Geological Survey, Alabama, Montgomery, Ala.

(18) *Publications*, California State Mining Bureau, Sacramento, Cal.

(19) *Reports* Aust. Assoc. Adv. Science Sydney, New South Wales.

(20) *Transactions and Proceedings*, New Zealand Inst., Wellington, New Zealand.

(21) *Quarterly Journal*, Geological Society, London.

(22) *Bulletins*, Inst. Mining and Metallurgy, London, E.C.

(23) *Transactions*, Inst. Min. Eng., London.

(24) *Monthly Bulletin*, Canadian Institute of Mining and Metallurgy, Montreal.

(25) *Journal*, Chem., Min., and Met. Soc. of S.A., Johannesburg, Transvaal.

(26) *Transactions*, Am. Inst. of Min. & Met. Eng. New York City.

(27) *Proceedings*, Colorado Scientific Soc., Denver, Col.

(28) *Journal*, Franklin Inst., Philadelphia, Pa.

(29) *Chemical Engineering and Mining Review.*

(30) *Transactions*, Am. Soc. C.E., New York City.

(31) *Bulletins*, Société des Ingénieurs Civils, Paris.

(32) *Mining Magazine*, 819 Salisbury House, London, E.C., mth., 1s.

(33) *Publications*, Iron and Steel Institute, London.

(34) *Proceedings*, Inst. of Mech. Eng., London.

(35) *Publications*, Field Columbian Museum, Chicago, U.S.A.

(36) *Journal*, Mining Society of Nova Scotia, Halifax, N.S.

(37) *Transactions*, Mining and Geological Institute of India, Calcutta.

(38) *Publications*, Department of Mines, Wellington, N.Z.

(39) *Journal of Industrial and Engineering Chemistry*, Easton, Pa.

(40) *Publications*, Bureau of Mines, Dept. of the Interior, Washington, U.S.A.

LIST OF ARTICLES.

GEOLOGICAL.

The Underground Geology of the Kolar Goldfield. T. Pryor. (22) No. 230.
The Tectonic Geology of the Sulitjelma Pyrite Deposits in Relation to Their Genesis. D. W. Bishopp. (22) No. 230.
The Gold Ores of Porcupine. J. E. Spurr. (5) Oct. 13, 1923.
Capping as a Guide to Copper Deposits. G. J. Mitchell. (5) Nov. 24, 1923.

MECHANICAL.

Emergency Power for Mines. Graham Bright. (25) Paper No. 1268, M.
Efficiency and Power Requirements of Fans. W. S. Weeks. (5) Sept. 22, 1923.
Steam Shovel Standardization. R. E. Tally. (5) Sept. 22, 1923.

METALLURGICAL.

Calculating the Zinc for Desilverizing Lead Bullion by the Parkes Process. G. G. Griswold. (25) Paper No. 1274, M.
Direct Electrolysis of Black Copper Anodes of High Nickel-Lead Content. M. H. Merriss. (25) Paper No. 1273, M.
Some Effects of Zirconium in Steel. A. L. Field. (25) Paper No. 1266, S.
A Contribution to the Kick *versus* Rittinger Dispute. H. E. T. Haultain. (25) Paper No. 1262, M.
The Concentration of Gold-Copper Ores by Froth Flotation at Tul Mi Chung, the Seoul Mining Co., Korea. R. J. Lemmon. (22) No. 229.
Geology and Metallurgical Treatment of the Sullivan Zinc Ore. C. R. Whittemore. (24) No. 136.
The Panda Copper Concentrator of the Union Miniere Du Haut Katanga. A. Simmons. (32) Sept., 1923.
Assaying and Metallurgical Analysis at the Falcon Mine, Southern Rhodesia. E. Golding. (25) Sept., 1923.
Rubber Lining for Tube Ball Mills. A. B. Parsons. (5) Sept. 22, 1923.
Possibilities of "Sand Flotation" of Ore in a High-Density Pulp. H. M. Chance. (5) Oct. 13, 1923.
Leaching a Copper Ore. F. E. Wormser. (5) Oct. 20, 1923.
Possibilities of Centrifugal Force in Ore Dressing. O. C. Ralston. (5) Oct. 20, 1923.
Mining, Concentrating, and Selling Zinc Ores in the Tri-State District. H. H. Wallower. (5) Nov. 3, 1923.
An Experimental Study of Ball-Mill Grinding. D. H. Fairchild. (5) Nov. 17, 1923.
Crushing 2,500 tons of Rock Per Shift at Corona, Calif. A. W. Allen. (5) Nov. 24, 1923.
Roasting Ores in Hydrogen. Hartwell Conder. (5) Dec. 1, 1923.

MINING.

Loading Ore Underground with Scrapers at the Utah-Apex Mine. S. P. Holt. (25) Paper No. 1267, M.
Mining Methods at Bawdwin Mine. A. B. Calhoun. (25) Paper No. 1258, M.
Mining Practice at Sterling Hill Mine, New Jersey Zinc Co. G. H. Salton. (24) No. 137.
Some Notes on the Application of Airscrews to Mine Ventilation. F. A. Steart. (25) Aug., 1923.
Recent Developments in Underground Mechanical Loading. C. E. van Barneveld. (5) Sept. 22, 1923.
Concrete Caisson Used to Sink Shaft Through Wet Sand. R. F. Mundorff. (5) Sept. 29, 1923.
Interconnection of Contiguous Mines. G. J. Young. (5) Sept. 29, 1923.
Fresnillo Glory-Hole Mining Practice. T. C. Baker. (5) Dec. 1, 1923.

MISCELLANEOUS.

Ground Movement and Subsidence. (25) Paper No. 1271, M.
Helium: A National Asset. R. B. Moore. (25) Paper No. 1269, M.

Results Obtained by Schlumber Method of Electrical Prospecting. S, F, Kelly. (25) Paper No. 126I, M.

Present Tendencies in Exploration of New Mines. A, Locke. (25) Paper No. 1260, M.

Tacheometry in Rough Country. C. G. Lanfear Thompson. (32) Sept., 1923.

Kangaroo Hills Tinfield, Queensland. W, J. Morley. (32) Oct., 1923,

Mineral Constituents of Rand Concentrates. R. A. Cooper. (25) Oct.. 1923.

Rescue Work on Mines. R. R. Sayers. (25) Oct., 1923.

How to Select Prospecting Equipment. G. J. Young. (5) Sept. 22, 1923.

Marketing of Slab Zinc and Zinc Concentrates. F. E. Wormser. (5) Sept. 29. 1923.

Papers and Discussions.

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Rock-Drill Steel Problem in Australia, with some suggested steps to its solution (with 19 Figs.). By G. L. Thirkell, B.SC. - - - - 79

An outline of the Economic Geology of Tasmania. By Loftus Hills, M.B.E., D.SC. - - - - - - - - - 129

High-grade Zinc in Industry. By Harry Hey - - - - - 149

The Handling of Materials at the Risdon Works of the Electrolytic Zinc Co. of Australasia Ltd. (with 9 Figs.). By A. W. Cook, B.SC. - - - 167

The Principles and Applications of Thermo-Electric Pyrometry (with 7 Figs.). By Leonard B. Haney - - - . - - - 185

The Uses of Cadmium. By H. C. Urquhart, M.SC. - - - - 207

The Electrostatic Precipitation of Suspended Particles from Gases (with 4 Figs.). By H. C. Urquhart, M.SC. and K. I. Marshall - - - 223

A Few Notes on the Hydro-electric Power of Tasmania (with 9 Figs.). By C. C. Halkyard, M.C.E. - - - - - - - - 241

The Action of Light on Cinnabar. By W. H. Cropp - - - - 259

Comparative Cost of Rock Drilling Underground. By Thos. G. Hanton - 267

The Hydrogen Roasting of Pyritic Ores with Recovery of Sulphur and Iron. By Hartwell Conder, M.A. (Camb.), A.R.S.M. (London) - - . 287

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VENTILATING METALLIFEROUS MINES: A METHOD OF ATTACK.

PART II

BY THOS. G. HANTON.

(*Read before the Broken Hill Branch on* 17*th October*, 1922.)

THE following really forms the second part of, or a sequel to, a paper, "Ventilating Metalliferous Mines: a Method of Attack," read before the Broken Hill Branch of the Australasian Institute of Mining and Metallurgy in May, 1920. In that paper the writer advocated a consideration of the question of ventilation in metal mines from a different aspect to that which had hitherto prevailed locally. Various factors regarded as having an important bearing on the subject were set forth, together with the results of observations made during ventilation tests carried out at the mining department of the Zinc Corporation.

During the subsequent discussion the question was raised of the possibility of any practical application of the method of attack described, and also whether the theoretical considerations mentioned were likely to be of any real utility to the mining engineer engaged in ventilation work in a metalliferous mine.

The present paper, besides giving an account of the development and completion of the Zinc Corporation ventilation scheme, will also afford members an opportunity of judging for themselves the extent to which the method of attack previously outlined is applicable in actual practice.

This paper, then, will necessarily treat more particularly of the ventilation problem as it affected the Zinc Corporation rather than of the more general aspect of the question.

There was a general resumption of underground operations in November, 1920, at the conclusion of the strike of workmen commenced in May, 1919. Since January, 1921, however, no stoping has been in progress.

Prior to the settlement of the strike in November, 1920, a detailed scheme, following as far as possible on the lines set out in the first paper, for the underground work necessary for the adequate ventilation of the mine, had been evolved. This work was put in hand as soon as the mining department resumed. From January, 1921, onwards a limited number of men have been engaged underground in carrying the scheme to completion. This portion of the work has been practically complete for some time, and now the whole main programme has been fulfilled with the permanent installation and running of a fan on the main upcast.

The ventilation position in May, 1920, was as follows :—The construction of a main upcast had been commenced in two sections—from the surface to the No. 2 level (the stripping of the old No. 4 shaft being in progress), and from No. 5 level to No. 6 level, the size of openings fixed upon being 13 ft. 4 in. by 6 ft. 10 in. Extending from No. 2 level to No. 3 level, and from No. 3 level to No. 4 level, through the filling of the depleted stopes, there was an opening of three square sets, each 4 ft. 2 in. by 5 ft. 2 in. in the clear. All the above are near the northern boundary.

A No. 8 Sirocco fan, rated at a maximum capacity of 40,000 cub. ft. per min. at 2 in. water gauge, was installed on the old No. 3 shaft. This provided a temporary main upcast above No. 2 level while the enlargement of No. 4 shaft was in progress. The remaining portion of the upcast was through various sandpasses. The main downcast was through Block 5 shaft, supplemented to some extent by the south end sandpasses.

At that time the behaviour of the main shaft was variable. Fig. 1 shows the location of the various shafts.

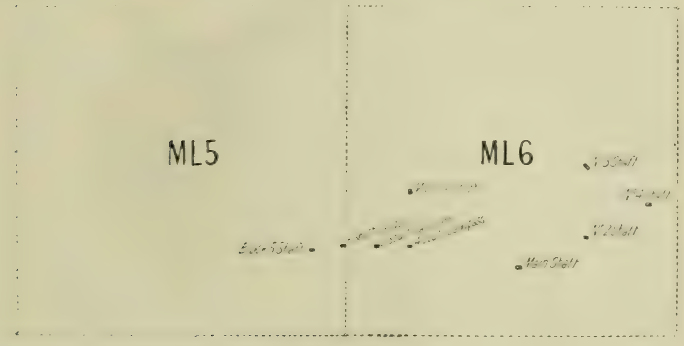


FIG. 1.
SCALE—1 IN. = 600 FT.

There was also on the mine a 119-in. diam. single-inlet Sirocco fan, which had been ordered for a capacity of 160,000 cub. ft. per min. at 4 in. water gauge, and rated at 150 b.h.p. To drive this fan a 225-h.p. synchronous motor had been bought, this being the nearest size after the necessary additions for belt friction and other contingencies had been made. Other considerations also entered into the matter of the type of motor adopted, the chief being the raising of the power factor for the whole plant.

In drawing up the ventilation programme, the following were regarded as being the main points for consideration :—

(1) The ventilating pressure, or water gauge, at which it would be desirable to work.
(2) The size of the main airways, especially that portion of the upcast through which the total mine air would pass.
(3) The volume of air required in circulation to adequately ventilate the whole of the workings.
(4) The type and capacity of the fan to be installed and the power necessary to drive it.
(5) The regulation of air supply to the various levels, and the distribution of air as between levels and stopes.

These features, of course, are all interdependent.

(1) As the power necessary to circulate any given quantity of air through the mine varies directly with the water gauge, it was decided to work upon as low a ventilating pressure as practicable. Immediate capital expenditure on increased size of airways was considered preferable, within limits, to continuous current expenditure on the increased power required to force an equivalent amount of air through smaller passages at a higher water gauge.

Further, at the higher pressures there would be increased difficulty in effectively stopping or regulating the air currents. Even during the tests trouble was encountered in sealing off any particular portion of the mine with a motive pressure of 2.5 lb. per sq. ft. or 0.48 in. water gauge acting. Also as a portion of the upcast, which would probably be constructed would be in openings through the sand filling in depleted stopes, a low pressure was expedient. All things being considered, a maximum of 2 in. water gauge was fixed as a ventilating pressure which, if possible, would not be exceeded.

(2) It was next necessary to ascertain the pressure required to overcome frictional resistance for various sizes of airways and various quantities of air passing through them.

It may be remembered that in the figures for frictional resistance supplied in the first paper, those of the upcast and downcast comprised by far the greatest portion of the total resistance of the mine circuit. The resistance of the levels or any series of stopes, for the smaller quantity of air traversing them, was comparatively very small. As stated at that time, the value of k, the coefficient of friction, as taken from Peele's Handbook, did not appear suitable for Block 5 shaft. It had also been found that the values of k, as given in Peele, on application to levels and stopes, gave a fair approximation when checked on various closed circuits of which Block 5 shaft did not form a portion. The tentative figure of $\frac{.4}{10^8}$ for the downcast coefficient was further verified under conditions of natural ventilation. In Fig. 2 the particulars are shown for one instance. The circuits of maximum resistance along various levels were taken as set out there, for a day when the motive forces, calculated from the observed mine temperatures and pressures, acting on each level, were as shown. The resistance of the measured quantities of air along these circuits was calculated. The motive pressure acting on any level should equal the sum of the frictional resistances of the downcast of the

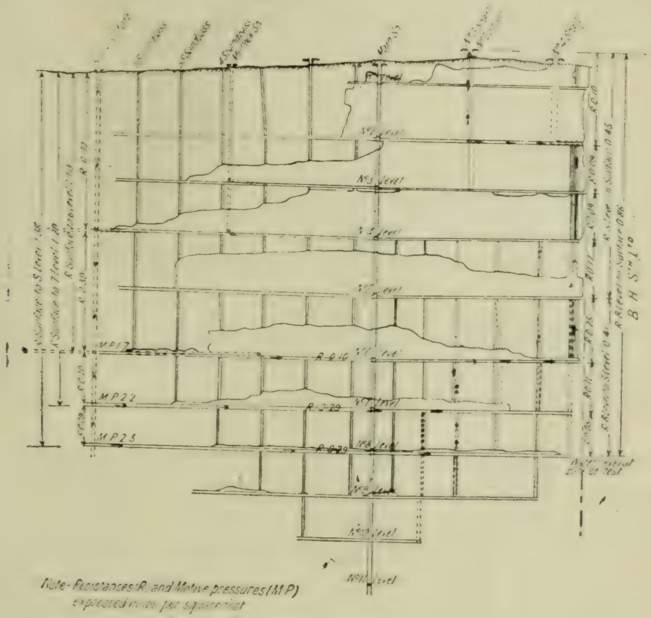

R Surface to 8 level 1.38
R Surface to 7 level 1.20
MP 17
MP 22
MP 23
R 0.29
R 0.29
No 3 level
No 8 level
No 9 level
No 10 level
No 11 level
RC 10
R 0.09
R 0.11
R 8 level to Surface 0.86
B H S No 10
Note - Resistances R and Motive pressures (MP) expressed in lbs per square foot

FIG. 2.

Longitudinal Section—Scale 1 in. = 600 ft.

level, and of the upcast(the path of maximum resistance being followed in each case) for the quantities of air in circulation in them due to this motive pressure.

Using $\frac{.4}{10^8}$ for the downcast friction coefficient, a fairly close agreement was obtained between the motive forces and the calculated resistances of the different circuits. These are as shown in Fig. 2.

For No. 8 level circuit.—Calculated motive pressure, 2.5 lb. per sq. ft.

	Lb. per sq. ft.
Block 5 shaft resistance, surface to No. 8 level ..	1.38
Resistance along No. 8 level	0.27
Upcast resistance, No. 8 level to surface	0.86
Total	2.51

For No. 7 level circuit.—Calculated motive pressure, 2.2 lb. per sq. ft.

	Lb. per sq. ft.
Block 5 shaft resistance, surface to No. 7 level	1.29
Resistance along No. 7 level	0.27
Upcast resistance, No. 7 level to surface	0.81
Total	2.37

For No. 6 level circuit.—Calculated motive pressure, 1.7 lb. per sq. ft.

	Lb. per sq. ft.
Block 5 shaft resistance, surface to No. 6 level	1.09
Resistance along No. 6 level	0.1
Upcast resistance, No. 6 level to surface	0.7
Total	1.89

The agreement between these motive pressures and resistances indicated that the tentative value of $\frac{.4}{10^8}$ for Block 5 shaft was within reasonable limits, and it was used in future work. It was also adopted in main shaft calculations.

Throughout the paper the values of k are for use with velocities expressed in feet per min. and pressure in pounds per sq. ft.

The friction coefficient having been determined for Block 5 shaft, it was now possible to proceed with the calculation of the mine resistance for given quantities of air and with various systems of airways, the chief consideration being the size of the main openings which would be necessary to accommodate a sufficient quantity of air without exceeding a total of 2 in. water gauge.

Many resistance figures were computed for various combinations of airway size and volumes of air in circulation. For a start particular attention was devoted to the upcast, especially to that section of it through which the total volume of air in the main circuit would pass. It was early apparent that the upcast would be sealed off as completely as possible from No. 4 level to the surface.

The main producing levels from No. 5 downwards only were taken into account in the allocation of air from the main circuit. The stopes north of the main shaft from No. 4 level upwards are practically depleted. It was considered that No. 4 level and the levels above it would be served by a subsidiary natural ventilation circuit, downcast air entering through the south end sandpasses and upcasting through the old No. 2 shaft, No. 3 shaft, and any north end sandpasses which happened to be open.

By thus isolating the main upcast from No. 4 level upwards it will be seen that that portion of it extending from the point where the last upcast air would enter below No. 4 level to the surface would have to carry the total volume of the main circuit.

A suitable size for this section then had to be determined, work having already been started in two portions in forming an upcast 13 ft. 4 in. by 6 ft. 10 in., it was decided to adhere to this, or an equivalent area, if possible. Resistance figures were computed for volumes ranging from 30,000 to 160,000 cub. ft. per min. These showed that in a main upcast of approximately 90 sq. ft. area, and following a route decided upon, 40,000 cub. ft. per min. could be circulated in this section at a pressure of .67 lb. per sq. ft., or, for 80,000 cub. ft. per min., 2.68 lb. per sq. ft. This indicated that very probably an average area of 90 sq. ft. would be sufficient for this top portion.

The remaining sections of the upcast were now considered, estimates being made of the probable proportion of air that would traverse each under various conditions. These estimates were based on resistance figures and also upon the measured volumes in these airways.

Attention was then directed to the downcast starting with Block 5 shaft. This shaft is vertical and close timbered, size 12 ft. by 4 ft. 6 in. in the clear, divided into three compartments—two for hauling, each 13.9 sq. ft. in area, and a pump and ladder-way compartment 25.6 sq. ft. in area. The first work apparent was the clearing of this latter compartment as much as possible, the solid ladder stages to be replaced with iron grids. Resistances were calculated for various volumes entering at the surface, and allotted as desired to the various levels for 30,000 cub. ft. per min. (10,000 cub. ft. in each compartment), and taking the resistance of one cage compartment as a measure of the pressure required in the shaft, the resistance from the surface to No. 8 level (1173 ft.) was 1.86 lb. per sq. ft. For 40,000 cub. ft. per min. the resistance would be 2.98 lb. per sq. ft., and for 80,000 cub. ft. per min. 11.92 lb. per sq. ft. It appeared, therefore, that there were, as previously stated in the first paper, limits to the carrying capacity of Block 5 shaft if 2 in. water gauge were not to be exceeded for the mine total. It was tentatively decided that 30,000 cub. ft. per min. would represent the maximum for Block 5 under these conditions.

Attention was given then to Nos. 4 south, 5 south, and 6 south sandpasses. These extend from the surface through the stopes and levels to Nos. 8, 7, and 6 levels respectively. As the average cross section of each of these is about 6 ft. x 4 ft., and one or more of them was liable to be out of commission as a downcast whilst sanding was in progress at any time, it seemed advisable not to rely on these for more than a total of 10,000 cub. ft. per min. at any time.

Attention was next turned to the main shaft as an addition to the downcast. The original idea had been to let the main shaft serve No. 9 level and those levels below it, as none of these had direct connection with Block 5 shaft. A consideration of the resistance figures showed that it would also serve the portions of Nos. 5, 6, 7, and 8 levels north of it, and would really become the main downcast. Block 5 shaft and the sandpasses would serve the narrower workings south of the main shaft, the air passing on to augment the supply to the more extensive workings in the north end, joining the main shaft circuit. This shaft is 14 ft. by 5 ft. inside the timber, divided into four compartments, two for skip haulage, one for an auxiliary cage, and one for a balance weight. The effective area of this last is so small that it was neglected.

The calculations showed that the pressure required to circulate an average volume of from 13,000 to 14,000 cub. ft. per min. down one of the skip compartments of the main shaft would be approximately the same as that required in Block 5 shaft for a total of 30,000 cub. ft. As the other two compartments, with the same pressure, would carry approximately 16,500 and 9,500 cub. ft. per min. each, the total average from top to bottom of the main shaft would be 33,000 cub. ft. per min. Further, as each level from No. 5 down drew off its supply from the shaft, the total quantity entering at the surface could be considerably in excess of 33,000 cub. ft. without increasing the pressure.

The upcast openings having been decided upon and a general downcast scheme formulated, the total mine resistance for different quantities of air was calculated, the total volume being split up between the various levels and ideal paths allocated.

It was considered that, provided a suitable total volume for the mine was arrived at, the regulation of the supply to the various levels and the distribution between levels and stopes could be attacked in detail.

Many calculations were made with various volumes and different distribution schemes, but the one finally resolved upon as being adequate is shown in Fig. 3, a total of 80,000 cub. ft. per min. distributed as indicated, the main distribution being based on resistance figures and measured volumes. Of this, 80,000 cub. ft., it will be seen that 30,000 cub. ft. was allotted to Block 5 shaft (14,000 to No. 7 level and 16,000 to No. 8 level), of which latter 6,000 cub. ft. went to No. 9 level), 10,000 cub. ft. was allotted the sandpasses, and 40,000 the main shaft.

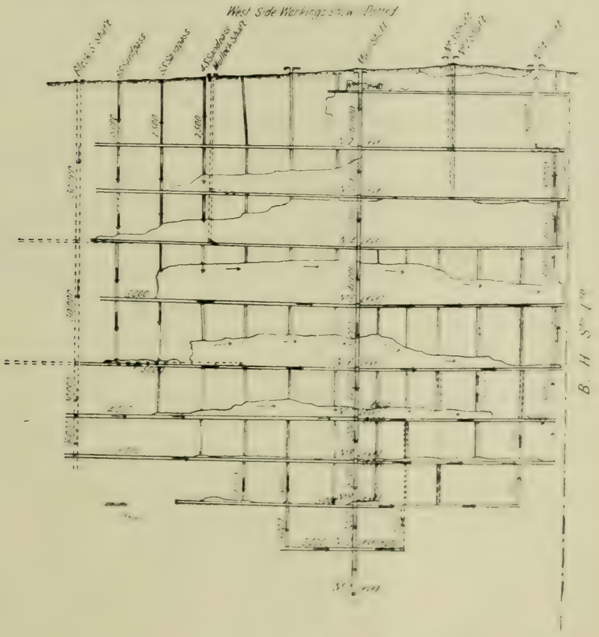


FIG. 3.

SHOWING PROPOSED MAIN DISTRIBUTION OF 80,000 CUB. FT. PER MIN. ENTERING THE MINE.

Longitudinal Section—Scale 1 in. = 600 ft.

Taking the mine circuit of maximum resistance, the pressure necessary to circulate the 80,000 cub. ft. per min., according to the distribution shown, was calculated at 1.6 in. water gauge. The above allocation was regarded as ideal, and it was considered that any alterations which would occur in practice, although the main quantities were about the same, would be in the direction of the air taking easier courses and splitting more, hence reducing the total water gauge.

In the above the loss of pressure due to changes of velocity over different parts of the circuit was not taken into account. In places such as the main upcast above No. 5 level, the loss of pressure due to a right-angle bend has an appreciable effect. With 80,000 cub. ft. per min. through a 13-ft. x 7-ft. airway, the loss of pressure in a right angle bend is 0.25 lb. per sq. ft. With two bends of 45°, under similar conditions, the pressure loss is .045 lb. per sq. ft. In these airways all turns were eased as much as possible.

The 80,000 cub. ft. per min. being under the maximum water gauge resolved upon, the next question was whether, when distributed as shown, it would be sufficient to adequately ventilate the mine. Detailed consideration indicated that, if the proposed allocation could be approximately adhered to, this quantity would be sufficient. This was considered again later, as will be mentioned. The scheme outlined was then adopted, and the work entailed was put in progress immediately opportunity offered.

No further investigation was carried out until the upcast from No. 8 level up was practically complete. The No. 8 Sirocco fan was then transferred to the main upcast (No. 4 shaft). The fan was only temporarily installed, and it was easily shifted. It was considered that the increased circulation would benefit the work then in progress, and it was hoped that the conclusions already arrived at might be further confirmed, although it was realized that the fan was not well suited to the mine openings. Readings were taken throughout the mine, both with the fan running and also under natural ventilation. With natural ventilation a circulation of about 23,000 cub. ft. per min. through the main upcast was obtained.

At that time the fan was driven by a 25-h.p. motor running at 10% overload. Under these conditions anemometer measurements on No. 2 level showed a circulation of approximately 30,000 cub. ft. per min.

Experiments were then made with this fan to test the effect of an evasé chimney on the outlet. Statements have been published that, with an evasé outlet, a greater volume of air can be circulated for the same h.p. or the same amount of air for less h.p. By the conversion of velocity pressure into static pressure, when the outlet velocity is reduced the back pressure on the fan is lessened, but the above statement appears to require qualification, as it depends on the suitability of the fan to the mine. The results were not very reliable, as the volumes all exceeded the fan capacity. When driven by a 50-h.p. motor and no chimney on, this fan circulated from 40,000 to 45,000 cub. ft. per min. Water-gauge readings in the side of the upcast shaft showed from 0.35 to 0.5 in. for these quantities. Readings in the fan duct itself varied from 1.7 to 2.5 in., indicating that the greater part of the fan power was used up in increasing the air velocity from that of the upcast shaft (470 ft. per min., or 7.8 ft. per sec. for 43,000 cub. ft. per min.) to that necessary to pass this volume through the fan duct, 3800 ft. per min., or 63 ft. per sec., the fan duct being 3 ft. 10 in. diam.

Thus :—

Pressure to overcome resistance of mine openings was .4 in
Pressure to increase velocity from 7.8 ft. per sec. to 63 ft. per sec. was .89 in.

the balance representing the resistance of the fan itself.

Taking the 0.4 in. water gauge read in No. 4 shaft for a circulation of 43,000 cub. ft. per min., the calculated resistance of the mine for a circulation of 80,000 cub. ft. was 1.4 in.

This corresponded closely enough with the calculated resistance for 80,000 cub. ft. to warrant the adoption of the latter. The 43,000 cub. ft. per min., while giving comparatively good results underground, was not considered sufficient for the mine requirements under working conditions, unless closer attention than would be possible or desirable were taken to ensure its adequate distribution.

The next consideration was the size and type of fan which it would be most economical to instal. The No. 8 Sirocco fan, as stated, was quite unsuitable, and, in addition, was almost worn out. It became a question of whether the 119-in. Sirocco fan already on the mine could be adapted or whether it would be desirable to instal some other type of fan.

Further inquiries from the fan manufacturers showed the following particulars :—

No. 68, 119 in. diam. Sirocco Mine Fan, with Evasé Chimney.

W.G.	Revs.	Single Outlet. E.O. = 32. Max. Vol. = 195,200.		Double Outlet. E.O. = 64. Max. Vol. = 390,400.	
		Volume.	B.h.p.	Volume.	B.h.p.
½	71	56,000	9	112,000	18
¾	87	68,800	16	137,600	32
1	100	80,000	21	160,000	42
1¼	112	88,800	30	177,600	60
1½	122	97,600	38½	195,200	77
1¾	132	105,600	46	211,200	92
2	141	112,900	54½	225,600	109
2¼	149	120,000	65	240,000	130
2½	157	126,400	76½	252,800	153
2¾	165	132,000	90	264,000	180
3	172	138,400	104	276,800	208
3¼	179	144,000	111	288,000	222
3½	186	149,600	118	299,200	236
3¾	193	154,400	131	308,800	262
4	199	160,000	144	320,000	288
4¼	205	164,800	162	329,600	324
4½	211	169,600	183	339,200	366
4¾	217	173,600	200	347,200	400
5	222	178,400	216	356,800	432
5¼	228	183,200	232	366,400	464
5½	238	187,200	248	374,400	496
5¾	239	191,200	277	382,400	554
6	244	195,200	306	390,400	612

(NOTE.—The above figures apply only in cases where the equivalent orifice of the mine is as given on this list, and where the upcast shaft and fan drifts are of suitable size.)

Quotes were received for other fans, together with particulars of their capacities.

As ventilating pressure varies as the square of the velocity, or volume of air circulating, and as this is also the law followed by flow through an orifice in a thin plate, the mine resistance may be expressed in terms of the area of an equivalent orifice, affording a basis of comparison for mine resistances. The closer the equivalent orifice of the fan approaches that of the mine the greater the efficiency will the fan give. If the equivalent orifice of the fan is less than that of the mine it will not circulate the quantities indicated.

From various observed volumes and water gauges the equivalent orifice of the mine was calculated. The following are examples :—

43,000 cub. ft. per min. at 0 .4in. W.G. $A = \frac{.0004 \times 43000}{\sqrt{4}} = 28.6$ sq. ft.

53,000 cub. ft. per min. at 0.5 in. W.G.,
with evasé chimney .. = 30.3 ,, ,,
47,628 cub. ft. per min. at 0.4 in. W.G. = 31.6 ,, ,,
Calculated 8,0000 at 1 ft. 6 in. W.G. A = 27.3 ,, ,,

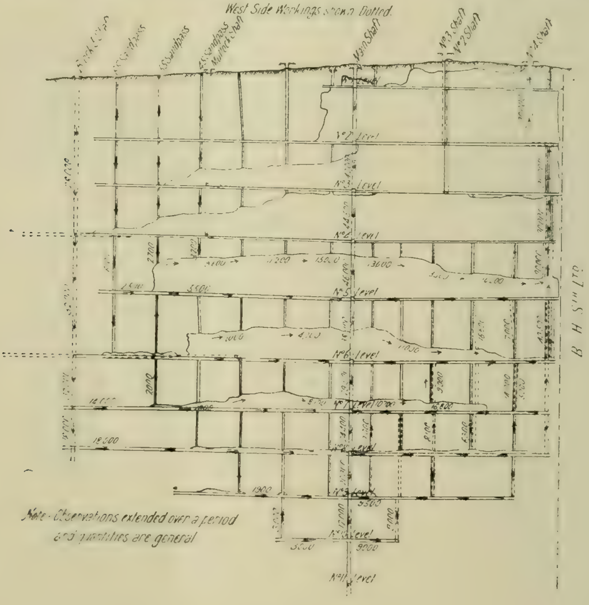
It thus appeared that the equivalent orifice corresponded sufficiently closely with the figure specified by the fan manufacturers for the 119-in. Sirocco fan.

Consideration of the problem showed that, after all, it would be more economical to instal the fan already on the mine, making the necessary alteration in pulleys to reduce the speed to supply approximately the quantity of air required. An important point in favour of the 119-in. fan was that, should it be necessary at any time to increase the volume of air through the mine, this could be effected fairly easily by an alteration of driving gear. Further, this fan, at the speed and h.p. indicated, showed an efficiency of 60 %, which was better than most of the other fans quoted for, where 80,000 cub. ft. was about the maximum capacity.

Consideration of the question by the engineering department showed that the fan could not well be run at less than 122 r.p.m., or a rated capacity of 97,600 cub. ft. per min. at 1.5 in. water gauge, taking 38½ b.h.p., said figures depending, as stated by the fan manufacturers, on whether the equivalent orifice of the mine were as per list.

It was therefore decided to instal this fan as being the best commercial proposition to run at 122 r.p.m. for 97,600 cub. ft. at 1.5 in. W.G., and rated at 38½ b.h.p.

Fan Testing.—The fan is connected to the canopy over the upcast shaft by a duct, 11 ft. diam. On account of the cost and constructional difficulties, it was decided to dispense with an adjustable shutter in the duct. Thus it was impossible to secure the necessary figures for a complete test or to plot the fan characteristics. A shutter was placed over the fan outlet to enable the fan to be started up at less than full load. On account, too, of the size of the duct, the average velocity for 90,000 cub. ft. per min. was too low to permit of the accurate absolute measurements by the Pitot tube of the volume passing through the fan, the manometer displacement being very slight. Anemometer readings were taken over the fan duct at twelve selected points, the duct being divided into three equal annular areas, and two readings taken both horizontally and vertically at the centres. These corresponded fairly closely with the measurements in No. 2 level ventilation crosscut 600 ft. north, a mean of eight readings being taken there.



Note - Observations extended over a period and quantities are general

FIG. 4.

SHOWING MAIN DISTRIBUTION WITH 119 IN DIAM. SIROCCO FAN CIRCULATING 90,000 CUB. FT. PER MIN. AT 1·6 IN. WATER GUAGE.

Longitudinal Section—Scale 1 in. = 600 ft.

With the outlet shutter closed the water-gauge reading was 0.3 in. above atmospheric, and the air in the duct revolved with the fan, leaving a dead area in the middle.

The manometric efficiency of the fan is calculated at 90.4 %, the theoretical water gauge for 90,000 cub. ft. being 1.77 in.

The average results were 90,000 cub. ft. per min., passing through at 1.6 in. water gauge. The engineering department observations show an expenditure of 53 h.p. at the motor.

The circulation and distribution is shown in Fig. 4. Although not quite according to the ideal mapped out, it is regarded as sufficiently good for all practical purposes. No regulating doors have been considered necessary. Since running the fan the only work required was additional doors for sealing off the upcast above No. 4 level.

At the present time the lower portion of the upcasts, on No. 7 level and below, are somewhat impeded, as ore broken in ventilation and development work underground has been stacked in the workings until milling operations are resumed. The clearing of these will give a better distribution.

The original scheme provided for the construction of a main upcast in east wall country-rock, connecting by crosscuts to the various levels and regulating the air supply for each level. A closer examination showed the possibility of this resulting in air going straight along the level only and into the upcast without giving full benefit to the stopes above.

It was decided that upcasts in the stope would be provided. and that in the lower levels these would serve for main upcasts also. At the same time mullocking facilities would be provided (a length of about 2 ft. has been subdivided and closely timbered off to carry sand filling, leaving the remaining portion always open for air).

With these connected to the main upcast circuit, a positive pressure is put on the stope, and the stope circulation improved. This has overcome the trouble mentioned in the first paper in forcing hot air down hill in the stopes ; the fan pull is sufficient to do this under these conditions. The draw is also upwards through the chutes from the level, and in places where the stope is restricted the quantity of air is made up by increased circulation on the free side of the stope. Openings of suitable size, about 8 ft. x 8 ft., are to be carried up under these upcast winzes as stoping proceeds, so that there will be no restriction of upcast area. Additional openings have been made from levels to stope to reduce the resistance where required. These will be provided for in all new work.

Most of the sandpasses between Nos. 4 south and 5 north from No. 6 level upwards have been blocked off to prevent short-circuiting. This was fairly easily done, as the openings were small, and it would not be difficult to keep them covered under working conditions. This was done in the early stages, but recent observations with the fan running have shown that it is not a matter of importance.

Finally, it may be stated that, in addition to the work described, much investigation work was done which either led nowhere or had no decisive results. In any future examination of ventilation questions this would be eliminated. Theoretical considerations were not regarded as important unless there was some possibility of their practical application. The immediate problem was the ventilation of the mining department of the Zinc Corporation. The only work described is that which had a bearing on that question, and the resultless portion has been omitted. Greater detail could be supplied, but would probably be very wearisome to those not closely acquainted with the mine. Although many of the results given and many of the details supplied may not be of very general interest, the development of the scheme from the stage attained at the date of the first paper to its completion now may prove interesting.

DISCUSSION.

Mr. P. H. Warren said that Mr. Hanton had contributed a paper which is a valuable addition to the records of the Institute. Although he stated that the paper did not treat of ventilation of metalliferous mines in a general way, but was concerned only with the particular problem of the Zinc Corporation, he (Mr. Warren) thought the author had shown that not only was there justification for the methods of attack adopted in the particular case mentioned, but that in no case should a ventilating fan be installed without an investigation designed to provide information relative to the total quantity of air necessary to adequately ventilate a mine, and the water gauge which might be consumed in doing so. In that connection he would like to ask if the volume upon which the calculations were based—80,000 cub. ft. per min.—took into consideration any future extension of the mine workings. Prof. Weekes, in a series of articles which appeared in the *Mining and Scientific Press* in 1920, dealt with that subject in a very interesting way, and also Walsh, in his text-book, " Physics and Chemistry of Mining and Mine Ventilation," clearly stated that in arriving at a basis upon which to design a fan necessary to do certain work, the steps should be—(1) to calculate the pressure necessary to pass the air through the main intake to the point where the first split branches off ; (2) calculate the pressure required to overcome the resistance of the main return airway, from the point where the last split of air was returned into it, to the fan ; (3) calculate the pressure necessary to pass the required volume of air through the hardest primary split. The sum of these three pressures would be the pressure required to overcome the total mine-resistance. Ventilation of metalliferous mines had been sadly neglected in the past, but, now that the importance of mine hygiene was realized, the subject was receiving much more attention, and, while it must be admitted that in many cases great improvement might be effected by the application of common-sense treatment, that was largely due to the fact that the field for improvement was so great ; and there could be no valid reason why the men engaged in that, as in other branches of a technical industry, should not be conversant with the scientific principles which underlay their work. Dealing now with the subject-matter of the paper. It was clear that the value of k for timbered shafts was very important, and Mr. Hanton seemed to have arrived at a value—viz., $\frac{.4}{10^8}$, which was not very different from those recommended by T. L. Elwen and Sarvaas. Perhaps Mr. Hanton would tell them if he regarded that value of k as suitable for general use when applied to the type of shaft commonly found on the Broken Hill field. It would also be interesting to know if he had given any thought to the value that should be assigned to k for air entrances to old square-set stopes ; these were often greatly constricted by staggered ladder stages, and sometimes contained two or three sharp bends. In discussing Block 5 shaft on page 9, reference was made to the necessity of clearing the ladderway compartment and replacing the solid ladder stages with iron grids, but he was not quite clear as to whether the figure of 11.92 lb. per sq. ft. necessary to circulate 80,000 cub. ft. of air per min. was calculated on the conditions obtaining in the shaft before or after the resistance had been decreased. It might be of interest to note that in the case of the main shaft on the Broken Hill South Ltd. they could not obtain a measurable quantity of air through the ladderway compartment till the solid ladder stages were replaced by iron grids, and a clear airway made through the shaft to the surface by the removal of the ladders to one end of the compartment; that alteration enabled them to obtain an additional 10,000 cub. ft. per min. When describing the fan tests Mr. Hanton said that difficulty was experienced in obtaining accurate velocity tests by means of the Pitot tube, as the manometer displacement was very small. Could not that difficulty have been overcome by the use of an inclined manometer giving a magnified reading? Anemometer readings could only be regarded as approximately accurate, even at speeds of over 100 ft. per min.; the slightest change in the angle of the blades would cause a variation in the accuracy of the reading. They usually obtain velocity readings in the fan passage at the South mine by means of the Pitot tube, but, as the velocity was about 2400 ft. per min., they had not found it necessary to use a magnifying manometer. On page 15 the manometric efficiency of the fan was given at 90.4 %, and, working out the theoretical water gauge by the formula $.00000115d^2v^2$—d being 119 in. (9.91 ft.) and v 122 r.p.m.—a result of 1.67 in. is obtained, as against Mr. Hanton's 1.77 in. In the above formula the relative density of water to air was taken at 893 to 1, a number of observations having provided them with that ratio. Possibly Mr. Hanton uses a slightly different factor, which would account for his theoretical water gauge of 1.77 in. It was stated that the average results obtained were 90,000 cub. ft. per min. at 1.6 in. of water. It was doubtful, in view of the fact that the fan could not be run with the duct closed, whether the above figures should be used in calculating a manometric efficiency figure. In any case, what value had the manometric efficiency ratio? Was not the over-all efficiency of the fan and motor of much more value for all practical purposes? On the figures supplied, the over-all efficiency of the fan and motor under discussion worked out at—

$$\frac{90{,}000 \text{ x } 1.6 \text{ x } 5.2}{33{,}000} = 22.7 \text{ h.p. in air.}$$

$$\frac{22.7}{53} \text{ x } 100 = 42.8\,\%.$$

The Zinc Corporation enjoyed an advantage, inasmuch as no doors were required for distributing the air through the workings. When attention was first directed to the ventilation problem on the South mine, they already had a Turbon suction fan capable of circulating 145,000 cub. ft. of air at a water gauge of 2 in., but, owing to the fact that both downcast and upcast airways were in the eastern country rock, there was a constant tendency to short-circuiting, which had to be corrected by placing resistance doors in the main drives and reducing resistances as much as possible through the stopes. They had engaged upon a policy of increased openings in all stopes between levels, an important item being the stripping of winzes. He gathered that Mr. Hanton also regarded the maintenance of openings in a vertical plane through the mine workings as a matter of prime importance, both from the point of view of distribution of air and the reduction of the mine resistance. Mr. Hanton had not supplied any information showing what improvements in the hygiene conditions of the mine resulted from the installation of the scheme. Such information would add to the value of a very able and interesting paper.

Mr. Hanton, in reply, said that Mr. Warren, in his discussion, brought forward some interesting points. Regarding the proposed circulation throughout the mine of 80,000 cub. ft. per min., distributed as shown in Fig. 3, it was considered that this volume would be sufficient for the mine requirements at a normal rate of extension of the workings. In Fig. 3 the allocation was given down to and inclusive of No. 10, the then deepest level. On the levels above No. 9 the major portion of the sill floor development had been completed. On Nos. 9 and 10 levels it appeared that, as extension took place, the volume of air entering would be sufficient, but instead of that air leaving them at the present upcast openings, it would upcast further north through openings which would be provided to keep pace with the other development work. Eventually the upcast would be at 600 ft. north, or at whatever points the northern continuation of commercial ore extended to on those levels. It was considered, then, that only the question of air supply for new levels as they were opened up needed to be taken into account. In the stage of mine development which had been reached it was thought that the opening of a new level would probably synchronize with the depletion of an old one. The latter would then be sealed off from the main upcast, and an amount of air equivalent to that previously traversing that level allocated to the new level. Thus, if the rates of development and depletion kept pace, as appeared probable, it was considered that the 80,000 cub. ft. per min. would be sufficient for future requirements. Actually, however, the fan was circulating 90,000 cub. ft. per min., so there was now a margin over the proposed 80,000 cub. ft. Also, as stated, with the present installation, should an increased circulation at any time become necessary, it could be effected fairly easily by a re-arrangement of the driving pulleys of the fan. That was not taken into account at the time. The series of articles by Professor Weeks to which Mr. Warren referred were very interesting. The first of those, published in the *Mining and Scientific Press* of April, 1920, unfortunately did not arrive at Broken Hill until after the reading of Part I. of the paper. Had it been available at that time it would have been of considerable assistance. Regarding the value of k, the coefficient of friction for shafts, the figure $\frac{.4}{10^8}$ obtained was considered suitable for "box" timbered shafts, the compartments of which had a mean radius in the neighbourhood of 1 or perhaps 1.5. For square or frame set shafts the value of k should be somewhat higher than in the case of close-timbered shafts. The question of a value of k for old square-set stopes did not arise here, as practically all ore-breaking was done in open flat-back stopes, square-set work being so far confined to the removal of the bridge of ore under a level. The friction figure would probably be high for square-set stopes, especially where there were obstructions such as ladder stages. It would be rather difficult in those cases to obtain more than a very general value. With reference to the pressure necessary to circulate 80,000 cub. ft. per min. through Block 5 shaft. 11.92 lb. per sq. ft., the calculation was made on the assumption that the ladderway compartment would be cleared as much as possible, the timbered stages being replaced by iron gratings. Allowance was necessary, however, even with that work complete, for a reduction of the effective area by air and water mains, electric cable, and the supports necessary for them. The pressure required for a given volume to pass through one cage compartment was always taken as a measure of the pressure in the shaft. The pressure in each compartment should be the same, the three compartments being in the nature of three splits extending from the surface to the first level, with an outlet from Block 5 shaft (in this case No. 7), and thence as splits from level to level. It may be mentioned that, in the calculation of k for that shaft, the effect of the ladderway obstructions did not enter, the volumes used being those actually measured in the one-cage compartment. Referring to the Pitot tube readings in the duct connecting the main upcast with the 119-in. diam. Sirocco fan, an inclined manometer with adjustable magnification was used, and as much as 5 to 1 was tried. The diameter of the duct, as stated, was 11 ft., and the area 95 sq. ft. That gives an average velocity of 947 ft. per min. over the duct for a volume of 90,000 cub. ft. per min. passing through it. Taking the weight of air at .07 lb. per cub. ft., the pressure created by that velocity was equal to .052 in. of water. That would give, with a magnification of 5, a displacement in the inclined manometer of .26 in. However, as shown by the anemometer readings in the duct, there was a variation in velocity at different points over its cross-sectional area. For those velocities at or higher than the average, fairly accurate readings on the inclined manometer were possible. With those much less than the average, it did not appear that reliable results could be obtained. The limit seemed to be about .20 in. inclined or 0.04 in. actual, corresponding to a velocity of 833 ft. per min. with air at 0.07 lb. per cub. ft. The magnitude of the error in reading the manometer would be of the same order, regardless of the total displacement. Hence, with a small displacement, the effect would be felt much more than with higher velocities, where the percentage error would be much smaller. As the complete set of readings could not be accurately obtained, the use of the Pitot tube was not continued. In the case of the

Broken Hill South mine, with a velocity of 2400 ft. per min. in the fan duct, the manometer displacement, he understood, would be in the region of 0.33 in. vertical. He was in complete agreement with

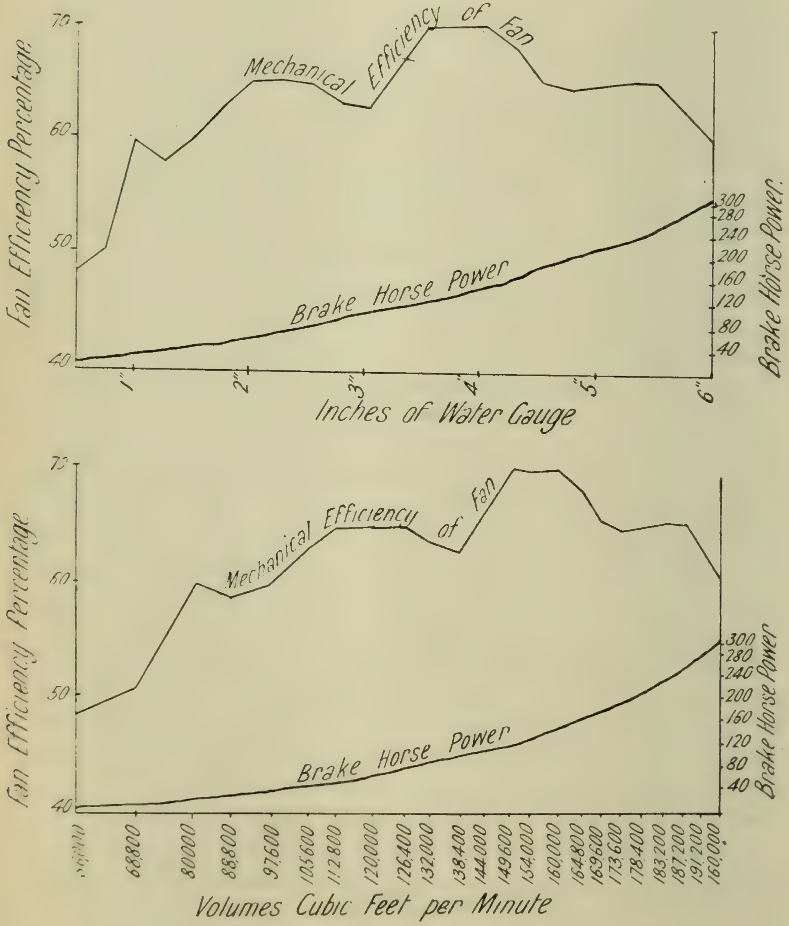


FIG. 5.

Mr. Warren regarding the disabilities of the anemometer. However, many sets of observations were taken on the No. 2 level ventilation crosscut 600 ft. north, connecting to the main upcast shaft, and in the fan duct—at eight points in the former and at twelve in the latter—and were regarded as being sufficiently close for all practical purposes. With regard to the manometric efficiency of the fan and the theoretical depression which it should produce, the discrepancy between the 1.67 in. calculated by Mr. Warren and the 1.77 in. given in the paper appeared to be accountable for by the difference in the figure used for the weight of air. Mr. Warren took .07 lb. per cub. ft., whereas the 1.77 was calculated upon a basis of .075 lb. per cub. ft., the latter just being taken as a general figure. The manometric efficiency was not regarded as being of much importance, the main point, as Mr. Warren has stated, being the over-all efficiency of the motor and fan installation. The 42.8% over-all efficiency mentioned by him agrees with their previous calculations. From the list supplied by the manufacturers, a copy of which was given in the paper, the efficiency curve of the fan in Fig. 5 was constructed. That shows that the fan was not operating under present conditions at the peak of the efficiency curve. With regard to the maintenance of airway openings through the vertical extent of the mine workings, those were considered essential to obtain the best distribution of air as between levels and stopes and also for the reduction of the mine resistance. In connection with Mr. Warren's question referring to the improvements effected to the mine hygiene as a result of the completion of the scheme, he would like to state that the temperatures observed had not been supplied, owing to the fact that the mine conditions were not at present normal. The sand filling throughout the stopes was now practically dry, and the moisture contents of the air comparatively very low. With the resumption of stoping and the restoration of normal working conditions the humidity would probably be considerably higher; therefore, any temperatures now given would hardly be a true reflex of actual working conditions. There was no reason, however, to think that they would be other than satisfactory. The volumes and distribution of air now in circulation, shown in Fig. 4, would afford a pretty fair indication of the present state of the mine. The circulation and volumes should not materially vary with a general resumption of work, and in places may, as stated in the paper, be improved. Mr. C. C. Freeman had raised the question of the order of the main points enumerated in drawing up the ventilation programme. The order, as set out, was not intended altogether to indicate their relative importance, as any variation in one intimately affected the remainder. Mr. Freeman was undoubtedly quite correct in one respect in saying that (3) " the volume of air required in circulation to adequately ventilate the whole of the workings " should be placed first in order of importance. The ventilating pressure, however, at which this circulation was to be attained was also a matter of prime importance, for upon that depended the power required for running the fan and the ease or otherwise of controlling the air currents. Upon the ventilating pressure and the volume of air in circulation would depend the airway size. If the pressure aimed at was so low that, for the volume of air required in circulation, an impracticable size of main airways would be necessary, then there would be no alternative to increasing the water gauge, so that the airway sizes for that fixed volume would be of practical size. The relation between ventilating pressure, volume of air in circulation, and airway size was so intimate that in any consideration of the problem it would be hard to differentiate between them.

SOME CALCULATIONS DEALING WITH MINE TIMBER.

BY H. F. PEARSON.

(*Read before the Broken Hill Branch on 7th November*, 1922.)

ONE of the phases of mining wide ore-bodies, such as are found in Broken Hill, is the laying-out and design of the sill floor gangway timber. When, in the above operation, a detail has to be repeated frequently, it is desirable in the interests of efficiency that that detail should be standardized as far as possible. Before arriving at a standard design—*e.g.*, a crosscut layout—it would seem necessary to compare its strength with some other unit of timber and see that the general design is at least as strong as the unit that has been taken for comparative purposes.

In these notes no attempt is made to calculate the stresses to which timber underground is subjected, but merely to draw a comparison between the relative strengths of the various cross sections of timber in common use underground.

It has been assumed for the purposes of comparison that the filling exerts pressure in a vertical plane only, and is uniformly distributed over any area. A beam supported at both ends, and with a uniformly distributed load, has the maximum bending moment at a point midway between the points of support, and equals—

$$\frac{WL^2}{8}$$

where

W = load per unit length.

L = distance between points of support.

The bending moment also equals $M\frac{I}{Y}$, where

M = modulus of rupture of the material in question.

I = moment of inertia of the section of the beam about a given axis.

Y = distance of extreme fibre from that axis.

The formula $\frac{WL^2}{8} = M\frac{I}{Y}$ only holds good until the elastic limit of the material comprising the beam is reached. When a timber beam of uniform section is loaded it is obvious that the fibres in compression will fail before the fibres in tension, because the tensile strength of timber is greater than the compressive strength. So the compressive strength of timber parallel to fibres has to be used in this case throughout as the modulus of rupture.

The moment of inertia of a square or rectangular section about a neutral axis that corresponds with the centre of gravity of the section is $\frac{b\,d^3}{12}$.

Where b = breadth of and d = depth of section, Y = distance of extreme fibre from the neutral axis and $= \frac{2}{d}$.

Stringybark is used underground on the Zinc Corporation, and, as the average compressive strength equals 7000 lb. per sq. in., that value has been taken. As a matter of fact, modulus of rupture could be dispensed with, because, provided only one variety of timber is used, the following calculations are simply based on L, b, and d of the above-mentioned formulæ. However, it is simpler to take a value for M.

The assumption has also been made by the author that the sill timber set is to be the unit of strength, and that no other combination of timber must be inferior to it in strength. Also, that the decking of a set should be the first member to fail.

The type of set in use on the Zinc Corporation was as follows :—Cap 5 ft. in clear and sill 6 ft. 4 in. ; legs 8 ft. 4 in. long ; sets 4 ft. centre to centre, with 10-in. x 4-in. decking resting on caps. Round logs had been used for some time, but, as it was decided to revert to square timber again, legs and caps of 10 in. x 10 in. will only be considered. We shall call this type I.

At the point of failing by compression in the compressive fibres, a piece of 10-in. x 4-in. decking, 38 in. between supports, will stand a maximum uniformly distributed load of

$$\frac{WL^2}{8} = \frac{MI}{Y} = \frac{W \times 38^2}{8} = 7000 \times \frac{10 \times 16}{6}$$

$$W = \frac{7000 \times 10 \times 16}{6} \times \frac{8}{38^2} = 1034 \text{ lb. per in. run.}$$

This equals a load per in. run on cap = 1034 x 4.8 = 4963 lb. per in. run.

The cap is 10 in. x 10 in. and 5 ft. in clear, and by working back we can find M that would be required to support this—

$$1034 \times 4.8 \times \frac{60 \times 60}{8} = M \times \frac{10 \times 100}{6}$$

$$M = 1034 \times 4.8 \times \frac{60 \times 60}{8} \times \frac{1000}{6} = 13{,}400 \text{ lb. per sq. in.}$$

But as extreme value for M = 7000 lb. per sq. in., it can be seen that the cap would fail before the decking developed its full strength. In such a set as this a cap 12½ in. x 12½ in. would be necessary, or a log 15 in. in diameter. It is obvious that where the length of cap is increased and the area to be supported increased also, the section of the cap must be enlarged accordingly. Besides increased cost of handling such large logs, there are other disadvantages to be considered. In a set that came under notice recently the cap was 8 ft. 5 in. in clear, and was supporting 6 ft. 9 in. of decking at one end, diminishing to 3 ft. 6 in. at the other end. Supposing that the maximum bending moment was that at the centre of the cap, and that the load was such that the decking on the drive sets was failing, this cap would require to be in the random of 20 in. square, or 24 in. diam. This log would be of a considerable weight.

It was then decided to investigate the question of placing longitudinal stringers from set to set vertically over legs and placing 10 in. x 4 in. decking on them. This will be called type II.

The length of decking would be 60 in. between supports. The maximum uniformly-distributed load would be—

$$\frac{W \times 60^2}{8} = \frac{7000 \times 10 \times 16}{6} = 415 \text{ lb. per in. run.}$$

Assuming 10 in. overlap on either side of point of support, the length of decking would be 80 in.

$$\text{Therefore, load on each stringer} = \frac{415 \times 80}{1022} = 1660 \text{ lb. per in. run.}$$

To find the necessary dimension of square stringers by working back we get—

$$\frac{1660 \times 38 \times 38}{8} - 7000 \times \frac{X^3}{6}$$

$$X^3 - 257 = 6\tfrac{1}{2} \text{ in. (nearly).}$$

An 8 in. x 8 in. stringer over this span would have a F.S. of 2, and would provide a more elastic member.

As the length of decking must at times be increased, the section must be increased also, as before mentioned. Taking, for example, a 10 in. x 6 in. and working back, we find that—

$$\frac{415 \times L^2}{8} = 7000 \times \frac{10 \times 36}{6}$$

$$L = 7 \text{ ft. } 5 \text{ in.}$$

—*i.e.*, a piece 7 ft. 5 in. between supports is as strong as a 10-in. x 4-in., 5 ft. between supports. Hence it follows that a 10 in. x 6 in. should never exceed 7 ft. 5 in. in the clear.

By working back on the same lines we find that an 8-in. x 8-in. stringer will support—

A span of 4 ft. 5 in. of 10-in. x 4-in. decking 5 ft. in the clear.
A span of 3 ft. 10 in. of 10-in. x 6-in. decking 7 ft. 5 in. in the clear.

A 10-in. x 10 in. stringer will support—

A span of 6 ft. 3 in. of 10-in. x 4-in. decking 5 ft. in the clear.
A span of 5 ft. 4 in. of 10-in. x 6-in. decking 7 ft. 5 in. in the clear.

It will be seen that, by working in various types of stringers and decking, the full strength of the timber as a whole can be made uniform, and the weight kept down to a minimum. Fairly large openings can be bridged over by the judicious placing of sets and stringers so that the opening will stand the same top weight as the ordinary set and yet keep the section of timber used within the limits of efficient handling.

According to Gordon's formula, an 8 ft. 4 in. — 10 in. x 10 in. should stand a load as a strut of 500,000 lb. A leg, however, is not a simple strut, but is subjected to transverse bending as well, and this greatly weakens the leg as a column.

In the set type I. the load transmitted to each leg by the cap = $\frac{5000 \times 80}{2}$ — 200,000 lb., so that leg would be sufficiently strong if a simple column. As the leg is inclined from the vertical there is an overturning moment that is counteracted by a force applied by the cap spreader or joggle to the top of the leg. This makes the reaction of the leg = 200,000 ÷ cos. 4° 45′ (inclination from the vertical) = 200,700 lb. The vertical pressure = 1034 lb. per in. run of 10-in. timber, therefore pressure normal to side laths = 1034 x sin. 4° 45′ = 35.6 lb. per in. run on the side lath, and in turn = 411 lb. per in. run on the legs, the 10-in. x 10-in. leg being capable of standing a load of 933 lb. per in. run. Under the load due to side laths of 411 lb. per in run, the deflection of the legs will be—

$$\frac{5WL^3}{384EI}$$

and equal $\frac{137}{384}$ in., using 1,800,000 for E.

The total bending moment on the leg is

$$200{,}700 \times \frac{137}{384} + \frac{411 \times 10{,}000}{8} = f \times \frac{1000}{6}$$

$$71{,}080 + 473{,}750 = f \times \frac{1000}{6}$$

$$f = \frac{544{,}830 \times 6}{1000} = 3269 \text{ lb. per sq. in. on extreme fibre.}$$

Add to this the stress $\frac{P}{A} = \frac{200{,}700}{100}$

and we have 5276 lb. per sq. in. compression on those fibres that are in compression through the bending of the leg. This is below the elastic limit of 7000 lb. in compression, but if the area of the decking supported by this set were increased 50 % the legs would be on the point of failing. This could be met by increasing the size of the legs if it were thought desirable.

Coming to the set type II., the load on one leg = $\frac{\text{N}415 \times 80 \times 4.8}{2}$ — 79,680 lb.—say 80,000.

Vertical force acting normal to side laths = 415 x sin 4° 45′ = 34.4.

So the load per in. run on leg = 34.4 x 4.8 — 165 lb.

This would create a deflection of $\frac{5WL^3}{384EI} = 1/7$ in.

Total bending moment = $80{,}000 \times 1/7 + \frac{165 \times 10{,}000}{8} = f \times \frac{1000}{6}$

Therefore, f = 1305 lb. per sq. in.

$$+ \frac{P}{A} = 800,$$

equals 2105 lb. per sq. in. compression.

This has an ample reserve of strength to allow an increase in the area of the decking per leg. On a comparison of these figures to those for stringers, etc., it might be said that the cross section of the leg could be diminished, or the distance, centre to centre, of the sets increased. But, if in wide stopes, the filling exerts a side pressure, due to that portion of the filling lying above the angle of repose, the legs would have another load on them due to this wedge. So a moderate factor of safety may not be out of place.

The Zinc Corporation has decided to revert to longitudinal stringers, and in the layout of sets to conform to the spans, etc., given previously. Whilst not advocating any particular design of timber set, the writer has endeavoured to show that by the application of a few well-known formulæ a more satisfactory arrangement of timber can be attained than by simply placing sets more or less haphazardly.

A layout may look quite satisfactory as regards truck lines, etc., but when compared to the standard set may reveal some otherwise unsuspected weakness in a position where repairs would be difficult and inconvenient. If the information derived from these formulæ should set the designer off on another track, and so lead to a more efficient and better-balanced structure, they will have justified their application.

Papers and Discussions.

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LEAD SMELTING AT SULPHIDE CORPORATION LTD.'S COCKLE CREEK WORKS.

BY H. A. EVANS.

THIS paper is not intended to be a detailed description of the lead-smelting plant at Cockle Creek, as this has been covered in previous papers for the Institute in 1918, Proceedings No. 31, by other members of the staff; but the idea is to give some of the practices and conditions peculiar to these works, also to give some experiments carried out and the conclusions arrived at. Unfortunately, owing to the closing down of the smelting and refinery plant at these works, some of the experiments could not be carried to a more definite and commercial basis, but they are given in the hope that they may be of interest to members, and may perhaps suggest further thought along the lines indicated.

ROASTING AND SINTERING.

The greater part of the ore treated at these works came from the Central mine, Broken Hill, but, in addition, a large quantity was purchased from other parts of New South Wales and from Tasmania. This purchased ore was seldom received in the form of a concentrate, but mainly in its crude state, and therefore had to be crushed. The crushing was done in a Krupp mill, and the ore then added to the roasting mixture. This addition of crude ore was one of the main reasons why the Huntington–Heberlein process was preferred at these works to the Dwight & Lloyd sintering plant, it being found that with so many ingredients to mix in, the rotary furnace, which was used as a pre-roaster, acted as a mixer in the first few rabbles of the furnace, doing away with the expensive mixing arrangements that would be necessary with the D. & L. machine. Also, on the D. & L. machine, in order to obtain a perfect sinter with this crude ore, it was found that it had to be crushed much finer than for the H.–H. pots. The rotary furnace was considered a very efficient mixer and pre-roaster, and the upkeep was very small. It was, however, considered that the fume loss was a fairly high factor with this furnace, but experiments were carried out with a view to lessening this loss. It was found that at certain spots the heat was intense, and that at these spots volatilization took place. Also, it was found that at these spots the SO_2 was very high, and showed no spare oxygen. Provision was being made to ensure a current of air passing more uniformly over the furnace, particularly over those parts where the heat was previously localized, it being hoped that by these better draughting arrangements sulphating and fume loss would be reduced, as the intense heat would be distributed. However, owing to the closing down of the smelting plant (brought about primarily by the strike at Broken Hill), this improvement could not be carried out as intended.

The ore was taken hot from the roasters and conveyed in trucks by electric locomotives to the H.–H. pots. The material was tipped into the pots, and when the pot was charged the hood was lowered into position and the blast turned on. This method works all right, and there is no need to damp the ore provided the material is not dusty, which means that the H.–H. mixture should not be too infusible a mixture, otherwise it will not partially clinker or ball up in the rotary furnace. The iron contents of the mixture were usually kept higher than the silica.

The capacity of the pot originally used at Cockle Creek was 2 tons, but this was later enlarged to 4 tons, and from experience at Cockle Creek it is considered that the latter size is quite large enough. Mixtures that would not work well in the 4-ton pot have been worked quite all right in the 2-ton pot. Another point is that if the size of the pot is increased beyond the 4 tons, a good deal more trouble is met with in the breaking of the final sinter.

The typical mixture of H.–H. charge was as follows :—

	%
Pb	45 to 50
SiO_2	7 to 10
FeO	7 to 14
CaO	7
M_2O	2 to 3
ZnO	10 to 12
S	14 to 16

In the final sinter the sulphur contents varied from 3 % to 4 %.

Advantage was taken of making use of the sulphur in the fumes. Tests were originally carried out, and it was found that the SO_2 contents of the gases coming from the pot varied considerably. Soon after the blast was turned on to a pot the SO_2 became higher, reaching as high as 12 %. As the pot gradually burned out the gas became lower. The average of all the pots, taking all the gases, was fairly low (under 3 %) ; but by eliminating the start and finish of the pot the tenor of the gas was brought up to a figure suitable for making sulphuric acid by the chamber process. The pots were arranged with an exit to the air and one leading to the sulphuric acid plant. Twenty-four 4-ton pots were placed in commission, the ore was in rotation charged hot into these, and the lid placed in position and luted up by a mixture of flue dust and clay. The blast was turned on, and at the beginning the gases were allowed to go to the exit until the pot had got well started, when the gas was turned into the flue leading to the chamber process. To keep things as regular as possible, tests were taken of the SO_2 every half-hour by the man in charge of the acid plant. If the gas were above or below a certain percentage (usually it was kept at 6 %) the pot house was at once notified, and the pot-house attendant either allowed more dilute gas to go in or cut off some of the pots which were either just beginning or finishing. Later, tests were made to see if use could be made of the dilute gases going to waste, and it was found that if a cyclic system of working were adopted, practically the whole of the gas going through the pots could be made use of. In carrying out these tests, instead of the gas low in SO_2 being allowed to go to waste as previously, it was drawn by a Roots blower through a coke filter and was then returned, together with any fresh air necessary, to the bottom of the pot. It was found that if this returned gas was above 1.25 % SO_2, the sintering of the pot was interfered with, but by keeping this waste gas down and not allowing it to rise above the figure mentioned there was no alteration in the final product. If this plant had continued operations this system of working would have been adopted, thus making use of a greater quantity of sulphur than previously. This operation is shown in Fig. 1.

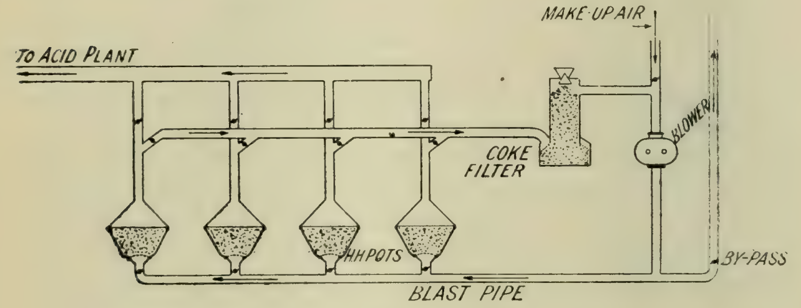

FIG. 1.

The coke filter used was a tower 4 ft. in diam., and contained coke broken to about 1-in. gauge. The gas entered at the bottom of the tower and was taken out at the top. From time to time the coke was taken out from the bottom and fresh coke added at the top, and by this means the dust was removed. The idea of this filter was to protect the blower from the dust that otherwise would have been sucked through.

During the eight weeks' run the blower was only cleaned once, and it was found that only a very little dust had carried through on to the beaters.

The sintered H.–H. material was carried by electric locomotive to the blast furnace and fed into hoppers, in which it was taken to the feeding floor of the blast furnaces.

The rotary furnaces were usually fired with coal, but, having a surplus of Mond gas available, this was brought over to the furnaces, and all the furnaces fired with this gas. Firing with gas reduced the fume loss to a certain extent, the probability being that not so much local heating took place.

Two D. & L. machines were installed, but were not used to any great extent, mainly for the reasons previously mentioned, but as pre-roasters they did good work. The material was damped and charged into the H.-H. pots, where a very excellent sinter was produced. It was found that damping the material gave a much better result, either

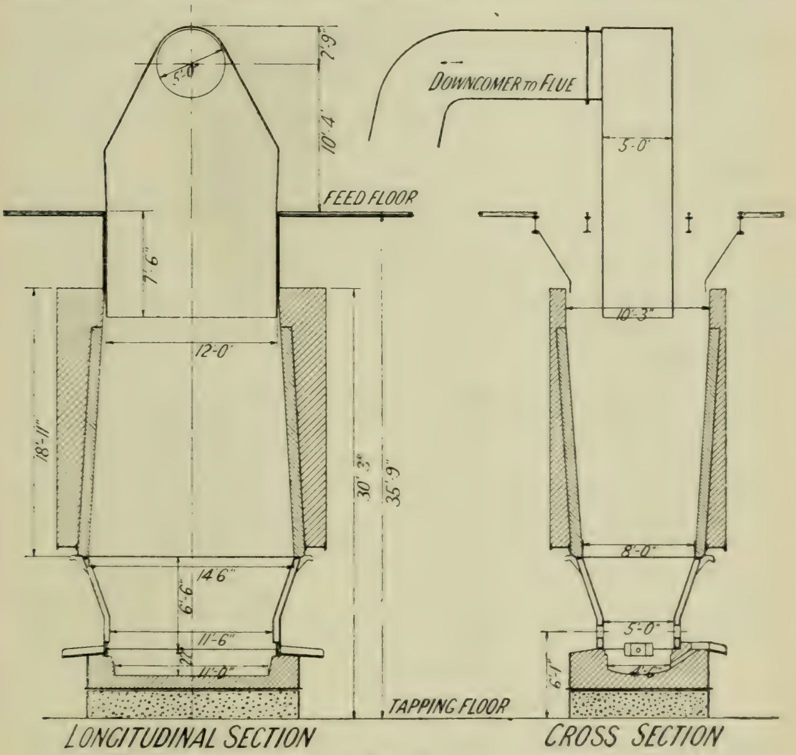

FIG. 2.
COCKLE CREEK SMELTER.

from the rotary furnace or the D. & L. From the rotary furnace the material was damped as it left the furnace, taken up to the H.-H. pots and started with a small fire. Another method of starting was to use hot material from the one rotary furnace, place a truck of this in the bottom of the pot, and the damp material on top. There is no doubt that the damping of the material gives a product lower in sulphur. This method was not continued owing to using of the gases for suphuric acid making. It was found that, using the damp material for acid making, the gases were very much lower in temperature, and did not work so well in the chamber process; also, when the lids were luted and the blast turned on to the damp material the hoods suffered severely, and the cost of renewing these hoods was considerable. Further, in making the acid, the CO_2 contents of gas from the starting up of the pots were detrimental. For this reason the damping of the material was not persisted in. The extra cost of handling, the upkeep of hoods, together with the disadvantages to acid making, were not counterbalanced by the slightly lower sulphur contents in the final product.

When the lead smelting was first started at Cockle Creek, the lead concentrates were hand roasted, and the resultant product went 5 % to 7 % sulphur. This, together with the necessary fluxes, was smelted in a blast furnace. A considerable amount of matte was produced, and metal recoveries were low. The fumes from the furnace were taken out at the side of the furnace just below the furnace floor.

Just after the installation of the H.–H. pots a new furnace was designed, where the flue was taken by a hood from the centre, the gases being removed by a down-comer. This was a great improvement. This second furnace had two rows of jackets, and the advantages of this furnace over the old type were considerable. Later on a new furnace was designed, a sketch of which is shown and dimensions given in Fig. 2.

The following are the principal dimensions of the different lead blast-furnaces built since the starting of lead smelting:—

Type.	Gases taken off.	Height from ground to feeding floor.	Height from tuyeres to feeding floor.	Inside width at tuyeres.	Inside length at tuyeres.	Size of bosh in jackets.
		ft. in.	ft. in.	ft. in.	ft. in.	in. ft. in.
1st	Side	17 9	14 0	5 0	10 10	5½ in 1 8
2nd	,,	21 9	18 0	3 6	11 0	10 in 2 8
3rd	Centre	34 0	28 0	5 0	11 0	8 in 5 0
4th	,,	35 9	29 8	5 0	11 6	18 in 4 6

It is considered that this last type of furnace does exceptionally good work. The blast used is from 55 oz. to 65 oz. The centre hood should extend into the shaft from 5 ft. to 8 ft. below top of feeding floor. A furnace of this size should have a bosh at jackets of

1 ft. 6 in. in 4 ft. 6 in. The jackets should be about 5 ft. high, and facilities should be provided for taking them out easily.

In the jackets of the present furnace the top originally had a cap or plate which was bolted down, and through this cap passed the inlet and outlet pipes for water. Later on this cap was taken off, and an open lip placed in position. A sketch of the jacket is shown in Fig. 3. The water was then taken by a hose pipe down to the bottom of the jacket. This open lip above referred to is much preferred to the bolted cap, as any pipes broken off or any incrustations occurring

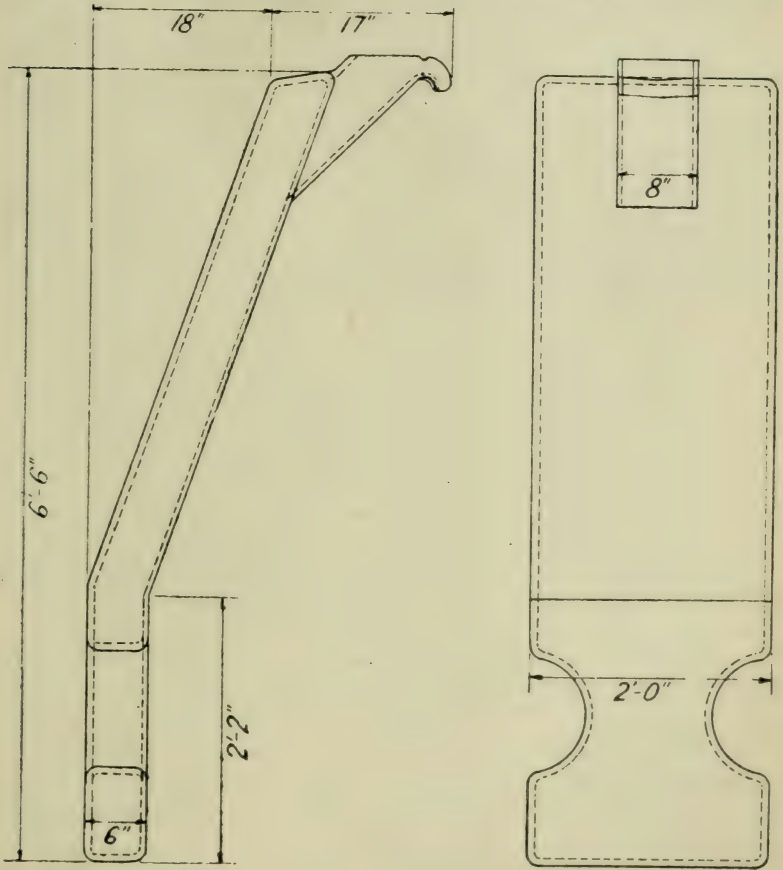

FIG. 3.

can be easily reached through the open lip. The walls of the shafts of the furnace should be taken straight up from top of jackets to the top of the furnace. The walls at the side should be taken as straight as possible, but in some cases a slight bevel is put in. In the third type of furnace erected both the sides and the ends had a considerable bevel, and in this furnace considerable trouble was experienced in scaffolding and crusting up. When the new furnace was designed the bevel was very small on the sides, but a larger amount was allowed on the ends to fit in with certain top arangements. This furnace showed considerable improvement with regard to " scaffolding," although a certain amount took place. Later on the ends were brought in, being made practically vertical—if anything, wider at the bottom than the top—and from that time no "scaffolding" and very little crusting up of the furnace took place. It was found that with a big slope the crusting of the furnace just above the jackets was very considerable, and that the straight sides were much to be preferred.

A slag that was usually run assayed :—

	%
Ag	.7
Pb	.7
FeO	35.0
MnO	1.5
CaO	17.0
SiO_2	24.5
ZnO	13.5 to 15

The amount of total charge smelted in the furnace (not including coke) for the 24 hours was 375 tons, consisting of :—

	%		%
H.-H. or D. & L. sinter	60	to	65
Fluxes (ironstone, limestone, carbonate ore, or quartz low in sulphur)	15	to	20
Returned slag	15	to	20

The coke consumption on the total charge (including returned slag) was 9 % to 10 %.

The lead in the H.-H. or D. & L. mixture went from 45 % to 50 %.

A great factor in the successful running of the furnace is speed. With slow running, troubles in the crucible are bound to occur, also scaffolding and crusting just above the top of the jackets.

In the writer's opinion, one of the chief factors in obtaining good speed is the quality of coke. A hard, dense coke does not work as well as one that has plenty of cellular space, and the quality of the coke has much to do with the decent running of the furnace. It was found that, using some of the hard, dense cokes, the furnaces decreased in speed, and considerable trouble was met with. With coke such as that made from a beehive oven the speed increased, and very little trouble was met with. This was more marked if the zinc contents of the slag were high. The disadvantage of making coke from Newcastle coals in beehive ovens is the low yield of coke, with no use of the by-products. The writer some years ago was shown some coke which was made with vertical retorts at a gasworks. The coal used was a mixture of Borehole and Maitland seams. The coke was hard and cellular. Trouble was experienced in getting the coke out of the retorts, as it was inclined to swell in coking. This difficulty, it was considered, could be overcome by alteration in design of the retort. Very likely, somewhat on the above lines, there is a possibility of producing a metallurgical coke, at the same time making full use of the by-products.

In connection with this matter the Company carried out a number of experiments with regard to the manufacture of tarless fuel by the low carbonization process, the idea being that the fuel produced after low carbonization would be suitable for use as a metallurgical coke and resemble that made from the beehive oven. This proposition appeared very attractive in view of the large amount of oils obtainable from the Maitland coals. However, the experiments did not produce a coke which was considered strong enough for lead-smelting purposes.

Experiments were then developed along the lines of the low carbonization of the coal heated by the gases coming from the Mond producers being allowed to circulate around or through the coal. This distilled off a certain amount of gas containing a considerable amount of tar. This tar containing about 75% oils. The porous coke remaining was then sent to the Mond gas producers and gasified, this gas in this way mixing with the gas produced from the low carbonization process.

By these means, caking coals after low carbonization, worked very well; whereas it was impossible otherwise to work them in the producers. A high yield of sulphate of ammonia (120 lb. to the ton of coal) would be extracted, together with tar, which on treating should give about 3 gal. crude naptha, 12 gal. fuel oil, and 10 gal. of pitch to the ton of coal.

These experiments were not continued in the face of the large increases in the prices of coal, wages and plant, which were not accompanied by a corresponding increase in the price of sulphate of ammonia.

The zinc contents of the slag at these works usually went from 13 % to 15 % ZnO; occasionally, higher slags were run. The chief thing to be considered in the running of the high ZnO slags is the sulphur contents of the material smelted. Provided the H.-H. or D. & L. sinter is low (3 % or under), no difficulty is encountered in running high zinc slags. As soon as the sulphur contents begin to creep up, trouble is met with.

For the purpose of running high zincs in slags and getting a low amount of lead in the slag, the chief factors as regards a lead smelter are a jacket with bosh of about 18 in., a high column with a downcomer to take the fumes from the centre, the furnace to be run as fast as possible, a good quality coke, and the sulphur contents of the H.-H. or D. & L. charge below 3 %. With these conditions no trouble should be experienced in running slags over 20 % ZnO and producing a slag going under 1 % lead.

The slag was tapped at both ends from this last type of furnace, and in front of each tap hole there was an iron fore-hearth. These were taken away at intervals and any matte or lead caught was tapped out and the crusts returned as returned slag. The lead was tapped in quantities of about 2 tons. It was found that by tapping the lead in this manner the crucible was kept in much better order than with the continual running of the lead, it being considered that, with the continual running, the dross is inclined to remain back in the furnace, whereas by a big rush of lead being let out from the furnace this dross is carried away. By this means no trouble was experienced with the crucible.

With ideal conditions obtaining in the furnace at Cockle Creek during a year's run, the crucible was never once interfered with, and barring down was very seldom resorted to—in fact, in the twelve months' run on the furnace, barring down only took place once.

The bullion was tipped direct into drossing furnace adjoining smelter and cooled. The first dross was taken off and the lead bullion moulded and weighed, placed in trucks, and taken to refinery. At the refinery a crane was used for lifting the lead bullion out of the trucks into an iron kettle in which the lead was melted and again drossed. The dross from this kettle, together with the dross taken off at the blast furnace, was sweated down to remove most of the free lead, and the dross then smelted in a small blast furnace to obtain the copper matte. This matte was at one time worked up to a blister copper, but later on was roasted and leached for making bluestone, the residue being returned to the H.-H. mixture· After drossing, the lead bullion was pumped to softeners , and then treated by the Parkes process. After the lead was softened it was run down into the desilverizing kettles, allowed to cool, and a further copper dross, which was added to the previous copper drosses taken from it, so that very little copper was contained in gold and silver crusts.

As a fair amount of gold was contained in the lead bullion, considerable difficulty was experienced in avoiding contamination of the silver crusts with the gold when they were taken off separately, and, although after taking off the gold, assays from the kettle showed that it was free of gold, but it was very often found that when the silver crusts were sampled after being torted they contained appreciable quantities of gold. It was therefore considered advisable to take off the gold and silver crusts at the same time.

After the copper drosses were taken off, the zincing from the previous charge, together with a certain amount of extra zinc (about 250 lb.) was added to the charge This brought the amount of silver down from 90 oz. to under 15 oz. A second zincing was then added to bring the charge down to under 0.15 oz. The last zincing was then returned to the next charge. The alloys were retorted, and the retort bullion cupelled. The Dore bullion was then taken to the parting plant, where it was treated in electrolytic cells for the separation of gold and silver.

In retorting bullion the cost of retorts and labour is high, and the same applies to the cupels. Running the latter is hot and tedious work, and fume losses are high. To overcome these objectionable features experiments were carried out on a scheme in which the alloys were to be first treated in an electric furnace, the resulting lead being then separated from the gold and silver by electrolytic methods. The first experiment was carried out in a small resistance type electric furnace, but this type of furnace was not considered satisfactory. It was thought that better results would be obtained with a furnace of the arc type. This furnace was being built when the closing down of smelting took place, and this part of the experiment was not carried further.

A small number of cells were built to treat the retorted lead by the well-known Betts process. The lead used in this experiment was that obtained from the ordinary retort furnace in the refinery. This small plant was capable of turning out a few cwt. lead per day. From these experiments a lead was produced assaying 99.5 %. A sludge was produced which went about 65 % to 70 % gold and silver, and 3 % to 8 % lead. Any antimony and arsenic that remained in the retort bullion was collected in the sludge. This sludge, after being dried, was taken and placed in a cupel with a small amount of soda ash as flux, and very quickly run up to a Dore bullion. The Dore bullion was cast into anodes and separated by the electrolytic method as used in the parting plant.

If the works had continued smelting, it was proposed to erect a larger experimental plant for treating the retort bullion, and, if this proved successful, enlarge it to treat the whole of the output from the retort furnaces, which, in the case of the Cockle Creek works, amounted to about 40 tons per week. It was considered that there was a possibility of a saving in labour costs and also in fume losses, besides which about 40 tons of electrolytic lead would be produced free of bismuth.

Fume and Slag Losses.

With regard to the losses of fume. A considerable number of tests were carried out in the different departments. It was often considered that a very large loss of fume took place from the H.–H. pots, but, on the gases from the pots being carried to the acid plant, it was found that the loss was very small. Dust chambers were placed in between the pots and the Glover tower, but the amount of dust collected in a year was so small that it hardly paid the cost of collecting. It could be definitely proved that very little fume was lost from the H.–H. pots, as, if it were not collected in the dust chambers, it should have been collected in the Glover tower or chambers ; but it was found that only a small proportion was caught in the Glover tower and none in the chambers. Tests were also carried out on the rotary furnace, blast furnace, and refinery.

With regard to the roaster gases, it was found that there was a fairly high loss of silver and lead from these, and the loss was greater when firing with coal than with gas, this being due to the local effect of the intense heat just beyond the bridge, caused by coal firing. However, owing to the large volume of dilute gases coming from these, it could not be seen how they could be profitably treated, and efforts were therefore made to avoid the volatilization at centre points on the roasters due to excessive localized heat.

With regard to the blast furnace, it was found that the losses were not so high as expected when the furnace was working well, and it was proved that the best form of flue-dust collector was the furnace itself.

Refinery losses were high with regard to the cupels, and it was considered that it would probably pay to have some means of collecting this fume.

The whole series of tests, however, proved that, from a commercial point of view, there was not such a great deal in collecting the fumes, providing the blast furnaces were working satisfactorily, and this fact should always be taken into consideration before installing any expensive dust-collecting apparatus. No doubt any cheap dust-collector that can be installed is quite legitimate; but it is very doubtful whether any of the more expensive installations which are often suggested have any chance of showing a profit after allowing for the cost of collecting, re-treatment of fume, etc.

In the determination of the volume two distinct methods were used. The first consisted of a series of Pitot-tube measurements of velocity. Pitot measurements were made at a number of places on the stack. The second method was based on the CO and CO_2 contents of the gases as determined by an Orsat apparatus. The amount of carbon passing into the gases in a given time was calculated from the amount of coke and limestone in the furnace charge, and, having found the proportion of CO and CO_2, the total volume of gas was calculated.

For the determination of the fume per unit volume, a sample of the flue gas was drawn through a 5-in. pipe by a fan from the centre of the stack and passed through a previously-weighed filter bag made of calico or flannel for a period of one to three hours. The bag was afterwards weighed and the contents assayed. The volume of gas filtered was found from the two distinct series of velocity measurements—one by Pitot tube and differential gauge in a length of 5-in. pipe leading to the filter, and another by an anemometer at the fan discharge.

Several other methods of calculating the fume were tried, such as inserting small tubes at various points of the stack and drawing the fume through them for a certain period; but it was considered that the use of the fan through a large pipe was much more satisfactory, and gave a truer idea of the amount of fume that it was possible to catch.

In the sampling of the slags dip samples were taken. It was very often found that when the furnace was showing signs of bad working the dip samples still showed low lead.

Experiments were carried out when the furnace was running badly, and dip samples were taken from the 3-ton slag pots. The slag pots were then taken and poured out, and the poured-out material, when cool, was sampled and assayed. The shell in the pots was treated in the same way. Both materials were put together and an average sample was assayed with the following result:—

	Ag.	Pb.	Cu.	FeO.	$CaCO_3$	Ins.	ZnO.	Al_2O_3.	MgO.	S.
Pot 1.										
Dip sample ..	1.4	1.8	.6	36.0	12.9	25.5	13.5	3.8	1.0	2.3
Bulk sample of slag poured from pot ..	4.5	3.9	.5	36.2	12.1	21.3	13.9	3.4	.9	5.7
Bulk sample of shell left in pot	6.5	5.5	.9	38.3	11.0	19.9	13.9	3.4	1.0	5.8
Average of pot..	4.8	4.1	.56	36.6	12.0	21.0	13.9	3.4	.9	5.7
Pot 2.										
Dip sample ..	1.0	2.3	.5	36.3	14.5	24.5	12.9	3.9	.9	2.5
Bulk sample of slag poured from pot ..	5.8	2.7	1.0	37.4	12.0	22.5	13.2	3.7	.8	4.1
Bulk sample of shell left in pot	8.8	5.3	1.2	38.8	10.7	19.0	13.9	4.0	.8	4.9
Average of pot..	6.2	3.1	1.0	37.6	11.8	22.0	13.3	3.7	.8	4.2

Whilst the furnace was running in very good order the same procedure was adopted, as follows:—

First Experiment—

	Ag.	Pb.
Dip sample	.3	.5
Bulk sample of slag poured from pot ..	.4	.6
Bulk sample of shell left in pot ..	.9	.9
Average assay of pot	.7	.8

Second Experiment—

	Ag.	Pb.
Dip sample	.5	.6
Bulk sample of slag poured from pot ..	.6	.8
Bulk sample of shell left in pot ..	1.0	1.0
Average assay of pot	.8	.9

From this it can be seen that, while the furnace is running badly, the dip sample does not convey a true idea of the amount of lead going out in the slag; also, that even when the furnace is running well, the dip sample is not to be entirely relied upon. Dip samples very often give a wrong idea of values in the slag dump.

From the tests carried out it was shown that, although the idea very often was that the greatest losses took place in the fume, in many cases there were greater losses in the slag than imagined, due to erroneous sampling, and it was shown that in this department, with greater care, the silver and lead losses could be reduced.

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NOTES ON THE LINDSAY REWARD TIN MINE, WESTERN TASMANIA.

BY D. V. ALLEN, B.Sc., A.O.S.M.

(Communicated by P. G. Morgan.)

WHEN the writer resided at Zeehan, on the West Coast of Tasmania, some years ago, he took a keen interest in the geology of the district, and availed himself of the opportunity to investigate occurrences of the varied minerals in which the field abounds. The following notes were compiled as a result of one of his periodical visits to the outlying mining districts, when an examination was made of the Lindsay Reward tin-mining property, which has an area of 140 acres. This is most easily reached from Renison Bell, the centre of the North Dundas tin-fields, on the Emu Bay railway, about eight miles north-east of Zeehan. Thence a pack track some 14 miles long connects with the mine, crossing the Pieman River by a fine suspension bridge. The mine itself is situated on the main south leading spur of the Parson's Hood Range, at an altitude of some 1600 ft. above sea level. The Parson's Hood Range is composed of granite, and is an extension of the rugged Meredith Range. A remarkable feature is the uniform topography of the surrounding country when viewed from the peaks. Except where dissected by the rivers and creeks, the surface is practically level, and, indeed, affords a striking illustration of an ancient uplifted peneplain.

The mine has been opened by means of trenches and adits, and the ore-body proved to extend for a length of 30 chains, with an average width of about 100 ft. Its general trend is west-north-west, and its dip, which is steep, corresponds to that of the country-rocks. These latter, which are of sedimentary origin and generally regarded as of pre-Silurian age, have been tilted at high angles, consequent, no doubt, on the crustal movements produced by the intrusion of the huge granitic masses which now form the high ranges in the vicinity. It may be here remarked that it is not within the scope of this paper to deal with the sequence of events leading to the geological structure as now existing in this district. That has since been very ably investigated by Mr. L. L. Waterhouse, of the Tasmanian Geological Survey. The original stratification of the rocks, which is very regular, has been remarkably well preserved. They are, for the most part, fine-grained slates, sandstones, and tuffs, considerably altered, however, by the proximity of plutonic masses, in places becoming extremely hard. The ore-body itself is striking in that it is stratified or banded in a similar manner to that of the enclosing country-rocks, the banding becoming the more noticeable as the walls are approached. This structure at once gives a clue to the mode of origin of the deposit, which will be discussed further on.

The adits have been driven approximately along the centre of the ore-body, with crosscuts at intervals to the walls, or, more correctly, to the boundaries of the mineralized channel. The lowest adit, some 20 ft. above the level of Tulloch Creek, which runs through the property, shows the bulk of the gangue to consist of a mixture of dark tourmaline and biotite, heavily impregnated with pyrite, with cassiterite or tin oxide fairly uniformly distributed throughout, and for the most part payable. Quartz is fairly common, occurring in small acicular crystals or as glassy blebs. Chalcopyrite, pyrrhotite, and arsenopyrite occur in minor quantities, while various secondary minerals resulting from their decomposition, such as malachite and siderite, are also found. Calcite, garnet, fluorite, and other minerals can be distinguished. Near the walls the sulphides give place to limonite, which is somewhat siliceous, and presents a gossanous appearance. The limonite in turn merges into a clay, and passes insensibly into the country-rock. The next adit, some 70 ft. higher, shows the ore-body to consist almost wholly of massive pyrrhotite (magnetic iron pyrites) with splashes of chalcopyrite, carrying cassiterite. The pyrrhotite occurs in bands of varying thickness, replacing layers of country-rock. The uppermost adit, after passing through a few feet of clay, penetrates the ore-body, here mainly massive magnetite, presumably of primary formation. Magnetite also occurs in defined bands, replacing the country-rock, and carries a little tinstone in the form of black crystals. At the summit of the hill, which is somewhat flat, a shallow shaft has been sunk, and a series of trenches cut across the cap of the deposit, here fully 100 ft. wide. The ore consists of magnetite and limonite, with scattered spangles of fine mica and clear blebs of quartz, the characteristic banded structure being well preserved. The western side of the hill slopes much more gently than the eastern side, and on it considerable quantities of oxidized material and clays have accumulated. An adit driven from this side of the hill carries limonite for most of its course, but when work was suspended pyrite had made its appearance in the face.

No extensive exposures of granite or its derivatives are observable on the Lindsay Reward property, there being, in fact, but two small outcrops of igneous rocks. One is a small tongue of altered quartz-tourmaline rock occurring close to the ore-body. The other and larger outcrop, in the form of a dyke, is situated some distance away, and on examination is seen to be a fairly coarse-grained holocrystalline rock consisting of quartz and feldspar, but devoid of mica, which may be called quartz-porphyry. In places clusters of black tourmaline crystals are present on the outside, having perhaps segregated to the margin by a process of differentiation when the dyke was forming. Although the igneous dykes are not directly connected with the deposition of minerals to any extent, indirectly their connection with the underlying massive granites is of great import, since it is from the latter that the metallic vapours and solutions giving rise to the deposits emanated. The huge granite mass forming the Meredith Range and its outlying spurs represents the acidic or final phase in the differentiation of the molten magma which intruded the older sedimentaries, and it was at this period that the tin was deposited.

As already indicated, the intrusion of these plutonic masses has had a marked effect on the sedimentary strata, metamorphosing them for a certain distance beyond the contact zone, and converting the sandstones into quartzites and the slates into cherts. The aureole of metamorphism, however, as far as can be judged, is not extensive, nor does the action appear to have been of a violent nature, since the rocks have not been shattered to any appreciable extent, and the bedding planes are still clearly discernible. Of more influence as a determining factor was the role played by the bedding planes, as they provided the channels along which the mineral vapours and solutions emanating from the cooling magma were conveyed. As the magma cooled the metalliferous vapours would first be given off. These, aided doubtless by superheated steam, extracted various metals from the magma and carried them into the channels afforded by the bedding planes. Their mineral contents would there be deposited, owing to reduction in pressure and temperature. Tin emanated as a volatile fluoride of tin, which at lower temperatures, in the presence of steam, was decomposed into oxide of tin (cassiterite) and hydrofluoric acid. The former was directly deposited in the parallel fissures or bedding planes, and so preserved the original banded structure of the strata. The hydrofluoric acid suffered a further reaction with the calcium compounds present, resulting in the deposition of fluorite. The association of these minerals (cassiterite and fluorspar) is characteristic of most tin fields. Quartz, which is almost invariably present, was simultaneously deposited by the action of superheated steam on the volatile compound, silicon fluoride. Although the cassiterite is regarded as having been deposited first, it is so intimately associated with the metallic sulphides as to suggest a practically contemporaneous origin. On the other hand, occasional rich veins of high-grade tin oxide are met with, which, since they cut obliquely across the sulphide bands, indicate that in such cases the introduction of the cassiterite is later than the deposition of the sulphides. It would appear that variations of this nature depend on the conditions prevailing at the time of mineralization. If from any cause these conditions alter, there is a corresponding change in the order of deposition of the minerals. When the conditions are intermediate the deposition of the tin oxide and the sulphides occurs simultaneously.

Metasomatic replacement has played a pronounced part in the formation of the ore-body, and adequately explains how whole bands of the country-rock have been completely replaced by mineral. The process can be seen in progressive stages, notably in the pyrrhotite bands. The action has proceeded simultaneously along a number of adjoining bedding planes, extending laterally until gradually the intervening layers are wholly replaced. The occurrence of tourmaline in the ore-body is interesting. Abundance of it can be seen occupying contraction cracks in the granite about the vicinity. The fissuring of the granite allowed the escape of boron vapours, which, reacting on the argillaceous sedimentary rocks, caused the formation of tourmaline. The effect of the various mineralizers extended some distance into the country-rock, and strictly the so-called walls of the ore-body were determined by the distance so penetrated. The absence of any considerable quantity of detrital mineral in the neighbourhood of the deposit may be accounted for in either of two ways —viz., that the mineralization of the ore-body did not extend to the surface, or that the steep nature of the country and the fineness of the tinstone were unfavourable for its accumulation.

Active operations on the Lindsay Reward property have now ceased altogether, but the former company, working in a small way, did much useful work in opening up and proving the ore-body. Sufficient was done to demonstrate that a large quantity of payable ore could be won from the oxidized zone. This, however, is small in comparison with the large reserves of sulphide ore from which ultimately the main supply would be drawn. But the proposition, to be profitable, requires handling on a large scale, with the expenditure of considerable capital. There is plenty of fall in the ground to allow of the mine being operated from a main adit and rises. Systematic sampling and assaying are essential. The method of ore-treatment calls for close investigation. Roasting would doubtless have to be resorted to at some stage of the process, and the possibilities of electromagnetic separation for certain portions of the ore should be considered. Cheap mining and milling costs would be all-important. By using the waters of Wilson's River to operate turbines, 1000 h.p. could readily be obtained under normal conditions. The power would be transmitted to the mine electrically, and would be ample for all purposes.

In a somewhat cursory examination of this complex ore-body the author came to the conclusion that there were many interesting problems demanding further elucidation, and of such economic importance as to warrant a detailed geological survey.

ROCK-DRILL DEVELOPMENTS.

BY THOS. G. HANTON.

THE following notes, compiled in the first instance for the makers of rock-drills, were read by Mr. Hanton at a meeting of the Broken Hill Branch on 12th September, 1922.

It was decided that all Broken Hill mines be asked to give their individual views on the subject, so that a comprehensive Broken Hill view could be published. Replies have been received from four (4) mines to date, and their ideas fairly well coincide with those set out by Mr. Hanton. His notes were as follow:—

"The Zinc Corporation Limited.

"NOTES ON ROCK-DRILL DEVELOPMENTS.

"As far as our present experience goes, the rock-drill developments on this mine will probably be something resembling the following. The programme is subject to modification or alteration, according to circumstances. The results of further trials and improvements in present drills, and the design of new types, will, of course, affect any general scheme such as that outlined.

"It appears reasonably certain that the 3¼-in. reciprocating drills, which have done practically all the stoping up to date, will be gradually replaced by hammer drills of the internal water-feed type, and that the existing types of hammer drill on the mine will be converted into internal water-feed machines instead of being used as dry machines with outside jet.

"The ultimate types of machines will be four.

"*Type* 1 would be a machine equivalent to the Ingersoll Leyner 248, Waugh Turbro, Sullivan DX61, or Holman CH2. This would be used for driving, cross-cutting, and similar excavating work, and possibly for stoping also, and should be capable of efficiently boring any class of ground. Features, apart from the general requirements set out later, would be the weight of the machine and mounting; height of the machine above the seating (as this affects the convenience of boring back or bottom holes); absence of projections on the mounting which the air hose will foul as the machine is fed forward; sufficient bearing surface in the chuck to prevent undue play of the steel, both in the hole and also in the shank end—the latter, of course, to prevent injury to the water tube; the ability of the machine to rotate a tight drill, and also to withdraw a drill which is tight in the drill hole. For the latter—*i.e.*, withdrawing a tight drill from a hole—it is very often convenient to keep the piston working while the drill is being withdrawn, and a machine which is cushioned in front should have advantages in this respect. Rotation independent of the piston action has some apparent advantages, and we think that this type of rotation could, with benefit, be further developed. he steel for these machines must be 1¼ in. round, with standard lugs.

"*Type* 2 would be a heavy jackhamer, weight about 60 lb., such as the Ingersoll DCRW13. The purpose of this is for winze sinking when used as a hand machine, and for stoping when mounted. An essential in a machine of this type is an effective cushioning device to prevent the vibration affecting the operator. A blow valve for directing live air through the steel at will for sludging the hole is desirable. On account of restricted underground operations we have not had an opportunity of trying out this type of machine in stoping, but our tests in other directions indicate the possibility of making this type of machine (relatively so much lighter and more convenient for handling in stopes than type 1) a standard for stoping when mounted on a convenient cradle. An arrangement whereby the cone may be made to slide on the shell appears to be a desirable feature in the mounting. Stoping is on the flat-back system, and the holes are approximately horizontal. The steel used in this type of machine is 1-in. hollow hexagon with 4¼-in. shank, which latter length we propose adopting as standard. The advantages of 1-in. steel over ⅞-in. for this work more than compensate for any decrease in drilling speed due to the greater diameter of the hole.

"*Type* 3 would be a lighter form of jackhamer than the above. Its purpose would be for 'popping' or boring holes in the large pieces broken down in the process of stoping. The early machines used for this were hand-rotated, such as the Waugh 22P. These have been superseded by jackhamers, such as the Ingersoll BCRW430, Sullivan DP33, Waugh Clipper, and Holman HD2, approximately 40 lb. in weight. These machines in the past were used for winzing, but in future the heavier machine, as per type 2, will probably be used entirely for that work. A lighter machine than 40 lb. would be effective for 'popping,' and when opportunity offers we propose to make a trial of machines weighing from, say, 25 to 30 lb., and having automatic rotation and internal water-feed. The decrease in weight would be an advantage in 'popping.'

"*Type* 4—Air-Feed Machines.—These are used solely for rising and a machine of this type, with internal water-feed, seems to be the most successful for rising under our conditions. We are acquainted with the Hydromax and similar types, but prefer the water-tube device to any form of anvil-block packing that has so far come under notice. We have not actually used this type (Hydromax, etc.) in rising, but tried it in driving. The leaks through the packing, and the difficulty of rotation in flat or depressed holes, renders it quite unsuitable, apart from other objections, for use on this mine Machines like the Ingersoll CCW11, with internal water-feed, are doing excellent work in rising. We see no advantages in automatic rotation combined with air-feed. In self-rotating air-feed machines we have tried the upkeep cost was excessive, and we are very doubtful whether a satisfactory machine combining the two features will be evolved. The steel used is plain 1-in. hollow hexagon. Essential features are a full and effective control of both feed and piston by one throttle only on the machine, and suitable seating for the water tube to prevent it working loose and dropping down into the rear portion of the machine. A desirable feature is one exhaust opening only on the front head (if any openings at all are left), so that the operator may keep on the side away from the exhaust. The easier of access the water tube is made the better, but it is difficult to see how a tube may be changed in an air-feed machine without partially dismantling the machine.

"*General.*—On all machines, except perhaps type 4, swivel air and water-feed connections are essential.

"*Columns.*—The proposed standard is 3½ in., as this will serve for both types 1 and 2 machines. For stoping the length would be 7 ft. Arms would be used in all cases.

"As regards the machines of all types, the desideratum, of course, is that combination of first cost, continuity of running, boring speed, upkeep cost, length of life, and air consumption, which will result in the lowest total cost per foot bored.

Regarding type 1, one mine replied "that development work in rhodonite, garnet sandstone, and very hard quartzite had not so far been accomplished as satisfactorily with these hammer machines as with a fast-hitting machine of the reciprocating type." The trouble may be attributable to a large extent to the failure of the drill steel, owing to the heavy duty entailed, and better results may be obtainable by using larger quantities of steel and changing at shorter intervals; but the writer is of opinion that the wear and tear on the machines will be excessive, and renewal expenses very costly in this class of ground.

There was a difference of opinion regarding the weight of a mounted jackhamer in type 2, three mines contending that the 46-lb. jackhamer was sufficiently heavy for all requirements.

The following general notes were also submitted in another contribution:—

" *General.*—The following general points covering design appear to us to be important:—

" (1) The drill should, in general design, have a stream-line appearance, and all projections eliminated as much as possible.

" (2) The machine should be of light weight compatible with strength. The English machines in type 2 are unnecessarily heavy.

" (3) The working parts of a machine should be reduced to a minimum, and for this reason we prefer the valveless machine. Our experience indicates that the rate of drilling is also greater with the valveless machine than with the valve machine.

" (4) All machines require frequent lubrication, and the introduction of an oil reservoir would help considerably in this respect, and at the same time reduce upkeep costs. All the drills, when boring back holes, require to be lowered before it is possible to oil them. This could be overcome by introducing an oil chamber of larger dimensions. We have found liquid grease better than oil for lubricating.

" (5) To prevent breakage of water tubes, the chuck-bushing should have a clearance not exceeding 1/64th in., and the chuck can be designed similar to the Sullivan DX61, which, in our opinion, is the best of any of the chucks in type 2 machines.

" (6) The independent rotation certainly has some advantages, and it is probable that further improvement will be made. We have not been able to obtain any greater drilling speed with machines having the independent rotation. In a machine having front–end rotation, but not independent, the number of working parts is reduced, but in the machine tested the rotation was not sufficiently rugged to withstand more than one month's work. This is being improved by the makers.

"(7) It is important that the machines should be cushioned at the rear end.

"(8) Air and water inlets should be of the swivel type, and the air entering at the centre of the rear end of the machine is an advantage."

VICE-PRESIDENT'S ADDRESS.

DEFENCE OF AUSTRALIA AND RELATION THERETO OF PRIMARY AND SECONDARY INDUSTRIES.

BY H. W. GEPP.

Mr. GEPP said :—" He proposed at the outset to draw attention to figures dealing with the population of the countries adjacent to Australia for the purpose of comparison with the population of the Commonwealth. He would deal more particularly with the areas of land surrounding the Pacific and Indian Oceans. Korea and Japan had a population of 78,000,000 ; China, 437,000,000 ; Persia, 9,000,000 ; Arabia, 5,000,000 ; South Africa, 129,000,000 ; Madagascar, 3,500,000 ; India, 251,000,000 ; Siam, Indo-China, and so forth, 16,225,000 ; Federated Malay States, 3,225,000 ; and, very significantly, Java, only four and three-quarter days from Fremantle, over 50,000,000. From the map displayed an idea of the comparative areas of the countries mentioned and Australia could be gained. It was rather amazing that Australia, with its great potentialities, had only about 5,750,000 people. New Guinea had over 750,000 people ; Hawaii, a mere speck, 250,000 ; New Zealand, 1,300,000 ; North America, 130,000,000 ; Central America, over 15,500,000 ; and South America, over 65,000,000. Colombo was only six days from Fremantle, and it was only nine days from Capetown to Albany, and eight days from Albany to Hobart.

Proceeding, Mr. Gepp said :—" It was in 1911 and 1912 that I had the good fortune to spend nearly twelve months in Europe, and during the course of my work in England and Germany to have the opportunity of studying both the economical and political systems in some detail.

" I was in Berlin at the time of the visit of Lord Haldane, when an effort was made by the British Government to arrive at some understanding with the controlling authorities of the German Empire looking towards the removal at least for the time being of some of the causes for the friction, which was becoming more and more serious, and which was threatening the peace of the world.

"I studied with great interest the amazing technical ability of the German nation, the assiduity of the German workmen, and the remarkable increase in the voting power of the so-called Socialistic parties in Germany, which, however, under the German constitution, gave them very little power in connection particularly with the foreign policy of the German Reich.

"I saw everywhere a tremendous activity in industry and in military preparations. In all parts of the country soldiers were drilling. On every drill ground evolutions were being steadily practised; the famous goose-step was part of the daily exercises of the conscripted youth of the nation, and the military official class was very evident. They carried things with a high hand, with a clear appreciation that they were the representatives of the ruling spirit of the bureaucracy which controlled the whole of Germany.

"In Belgium and France the manifestations of military activity were far less evident; the tone of the communities in those countries was much more subdued, and much less self-assertive.

"In Great Britain, where internal problems of serious magnitude were completely occupying the minds of the great mass of people, practically no evidences of military activity were apparent. I visited Weymouth to watch the review of the fleet, which, whilst very impressive, did not carry any such significance to my mind as did the much more obvious military preparations on the continent.

KRUPP.

"No huge arsenals or munition factories such as Krupp's, in Germany, existed in Great Britain. In Great Britain industries seemed to be mainly concerned in the manufacture of articles of commerce. A visit to the Krupp factories at Essen and Magdeburg showed clearly that a considerable majority of the tens of thousands of employees were engaged in the manufacture of weapons of war.

"When I visited Essen I was met by representatives of the firm and driven by coachmen in livery to the Essen Hotel, was conveyed from one portion of the huge factory to another by motor car, and when I asked for my bill at the termination of my stay was informed by the head waiter, with the click of the heels and with a profound bow, that I had been the guest of the House of Krupp. . .

" In passing, I would mention that one of the most significant facts at Essen was the care and attention given by the management to the question of cheap food, housing, and good sanitation.

" In Great Britain no special interest was shown in a traveller, his antecedents, or his business ; but in Germany every traveller was compelled to supply considerable information regarding himself and his intentions to the hotel proprietor, who duly reported them to the police, who doubtless thereby followed the movements in Germany of all foreigners and other possible suspects.

THE GERMAN PEOPLE.

" In spite of all the evidence so apparent, the imminent danger of a European war did not occur to me in any clear and unmistakable way. I felt that the complexity of modern trade and the fearsome results certain to follow upon a European war of any magnitude would prevent any serious trouble. I did not appreciate until after the war had broken out that a man or government cannot continue to prepare any weapon for a certain purpose—or, in other words, cannot continue to sharpen a knife to carry out certain operations—without sooner or later being tempted by some opportunity to try the efficiency of such preparations. It did not occur to me that the growing fear of the German people as a whole —that finally they would be squeezed between Russia and France and be attacked by both these nations simultaneously—placed in the hands of the German bureaucracy (with full control of the purse and the press) the power for evil which must sooner or later cause a gigantic and catastrophic upheaval.

" The German nation, highly skilled and thoroughly organized, and with a wonderful record of materialistic achievement for forty years prior to 1914, with an ever-increasing population, and persuaded by the evidence of their progress that they were of superior physical and mental calibre to the other peoples of Europe, grew more and more restless with their position in Europe. The fact that they did not possess any colonies in the temperate parts of the world upon which to build a world empire constantly vexed them.

A Place in the Sun.

"There were over 60,000,000 Germans crowded into a portion of Europe which could not continue to provide the means of livelihood to a largely increased number. The consequence was that emigration from Germany to portions of the world not under German control steadily increased. And, although the German Government refused to release the emigrants from their duties to the Fatherland, it was obvious that a great number of them would refuse to recognize these regulations, and were therefore lost to Germany in case of trouble.

"Consequently, there arose the insistent demand for what the Germans called 'their place in the sun,' and, to support this demand, the German Government, backed by the German people, started on their mad career of building a fleet to challenge the supremacy of the British.

Food Supply the Cause of Migration.

"Let us now turn for a few minutes to the consideration of the huge emigrations of people which have occurred at intervals during the history of mankind. We will realize very quickly that people swarming off from time to time from the parent stock have moved towards those parts of the world where fertile nature was able to provide food and general comforts. The various invasions by virile peoples of the valley of the Nile, of the valleys of the Tigris and the Euphrates, of the plains of Europe and of India, are clear illustrations of this fact, and these swarms of people have either destroyed or absorbed the previous inhabitants of these areas and built up a fresh civilization which has existed until they, in due course, have again been invaded by other swarms from over-popluated and less fertile areas.

"Another definite illustration is the various invasions of the British Isles by hardy Norsemen and other races from Scandinavia and from the adjoining portions of the Continent of Europe.

"When these various peoples settled down in the areas in which food was easily won (and the history of mankind is, after all, mainly a fight for food and for the areas where food is cheaply and easily produced), they found time to develop the arts and sciences and to increase the amenities of life. As these increased, the fibre of these peoples in all cases steadily decreased, until evidence became apparent to the hungry and hardy inhabitants of less favoured lands that the peoples in control of these rich areas were unable to defend themselves and were not recognizing the dangers of attack from outside —were, in fact, indirectly inviting such an attack, with every prospect of easy success.

AUSTRALIA'S POSITION.

" If we now turn to a consideration of our own land of Australia and consider its position in the light of the facts set out in the foregoing paragraphs, which facts can be paralleled from history many times without difficulty, we must feel grave concern ni regard to possibilities that within the next century Australia may have to fight for its existence as a nation.

" There are hopes that the steady progress of international comity may reduce the danger of war and of invasions, but to my mind the arguments that such hopes are sufficient to render unnecessary any preparedness for defence of such a fertile and unpopulated country as Australia are much too weak to carry conviction. I fear that human nature is unlikely to be so changed within the next hundred years as to prevent attack by over-populated countries upon fertile and unpopulated areas of the world, if the chances of success are even reasonably possible. Particularly is this so if, due to internal troubles or the pressure of internal politics, the condition of affairs invites the controlling portion of the population to look for relief in territorial expansion.

" I am not proposing to-night to discuss in detail the question as to whether Australia in the future may need to fight its battles of defence in some country other than Australia. I am not competent to discuss such matters from an expert's standpoint, and, indeed, for the purposes of my argument in this paper, such discussions are unnecessary. I am only concerned with the broad question regarding the steps (if any) which should be taken by Australia, in conjunction with such other portions of the Empire as are willing and able and feel called upon to assist for the purpose of preparing against attack and invasion at some future period.

NEED FOR PREPAREDNESS.

" A few months ago I gave a private address in Hobart, setting out my views on the steps which should be taken by Australia to prepare herself for defence against attack within 50 years from now. I propose to-night briefly to discuss this subject, and would say at the outset that, whether the period of preparation allowed be 50 years or 100 years, in my view it is essential that Australia should, to the limit of her financial resources and organizing ability, prepare herself to avoid what, in the absence of such preparedness, is inevitable.

"The richer the country and the more easily the needs of existence are won the greater the danger of national decadence. All the preaching in the world, all the warnings offered by the pessimists of the nation, all the teachings of history, and all the lightning flashes on the horizon of the political atmosphere, pass practically unnoticed by a wealthy nation unless the fibre of the nation be maintained by discipline and by the courageous and far-seeing and probably unpopular actions of parliaments and governments controlling its destinies.

"We need not for a moment discuss the directions from which danger may arise to Australia's integrity within the next hundred years. We know that nations change in their intrinsic composition. We know that the attitude of mind of one generation of a nation towards another nation may not be the attitude of mind of the succeeding generations. We may even postulate that the present social system and the present economic system may not continue to exist, and that the reasons for support or for attack may either increase or decrease until the foundations and the buttresses of belief in regard to international relations may be so undermined as to collapse and be useless, if not, indeed, to bring ruin with their fall.

THE START SHOULD BE NOW.

"It may be suggested that Australia, being a young nation, requiring all her energy and available wealth for national development and social amelioration, can afford to wait and to delay the necessary steps towards preparedness until it has sufficient population to make the burden easy to bear. My answer to this is that no date is too early to start the work for preparedness. No date is too early to commence to build an organization based upon such sound principles as to be generally applicable to every circumstance of international relationships. And if we, as trustees for the future generation which will inhabit this fertile land of Australia, have insufficient prescience at least to begin these preparations, then the future generations will doubtless never have reason to bless our memory.

"If we believe that the generations to follow us will be worthy of us, then surely it is our duty to make preparations which can be continued by posterity, and to which posterity is entitled.

Science and War.

"The terrific rate of advancement in the knowledge of the control and application of natural forces during the nineteenth and twentieth centuries has completely revolutionized the art of warfare, and, as it appears to me, has made the problems of defence of such an island as Australia much more possible of solution than would appear at first glance.

"Without posing for a moment as an expert in naval and military matters, and wishing you to understand that I only submit certain possibilities for the purpose of dealing with the fundamental ideas associated with the title of my paper, I suggest that the effective defence of Australia will call primarily for aerial and submarine units, and the building up in Australia of the necessary personnel and the supply of the necessary equipment for these services, and for a small but highly mobile, efficient, and thoroughly-equipped military force.

"The huge distances in Australia, which will still be of such importance even with a population of 50,000,000 people, are a predominating, if not a controlling, factor in the strategy of defence. But whether the land force and the naval force (exclusive of submarines) be large or small, the general principles of preparedness will apply. Any inability of mine, therefore, to discuss the strategy of defence is not seriously detrimental to the purpose of this paper.

Australia's Resources.

"We may take it that Australia can supply within her own borders almost every necessity for the maintenance and support of human life, and for defence as we know it to-day. But at the present stage of development there are certain vital products which Australia has only in the raw material form, and it seems definitely advisable for this generation to consider clearly what steps it can take towards the ultimate production in Australia of all necessary commodities—necessary particularly for her military and naval defence.

" Before proceeding with this portion of my subject, it need only be mentioned in passing that the basic essential more definitely necessary than any other is a sound, healthy nation with vision and clarity of thought, and with intrinsic community of interest, however many be the smaller differences of opinion between various sections of the community.

" Apart from the religious aspect—upon which it is not my province to touch— it is clear that physical and mental well-being of the nation is dependent, firstly, upon prevention of deterioration of the magnificent Anglo-Saxon stock from which this nation is descended, and, secondly, the provision and the steady pursuance of all methods to improve the nation, mental and physical.

" There is no time to-night more than to mention the necessity for steady health propaganda and action for the realization of the ordinary rules of health by all sections of the community, including, primarily, the care of the teeth. And in this connection I look with considerable hope towards the operations of the new Federal Health Department, which are so significant, and will be more so as members of that Department widen the scope of their assistance to the State and municipal authorities responsible for the general health of the nation.

" Australia can supply foodstuffs in plenty. She has ample sunshine, and in due course will spread her available water to all the areas of almost perpetual sunshine where growth is quickest and results are most secure. The science and art of the application of fertilizers and of agriculture generally to increase the productivity of the soil steadily widen, and even if the supplies of magnificent phosphate rock of the Pacific islands be temporarily unavailable, there are sufficient medium-grade deposits in Australia to tide over the difficulties.

" Of nitrogenous fertilizers there is ample at the moment. Doubtless in due course the artificial production of nitrogen-bearing materials will be undertaken in Australia for her own security and profit.

" Fuel oil and high-grade spirit for internal combustion engines are essential, and in this direction in the years to come every effort must be made to produce in Australia at least sufficient for the necessities of defence. Oil has not yet been discovered, but search is actively progressing, and meanwhile the Broken Hill Pty. Co., at its steel works in Newcastle, is successfully producing 25,000 gal. of benzol weekly, incidentally nearly 3% of Australia's consumption. Doubtless in due course production of similar materials from the magnificent high volatile coals of New South Wales and from the shale oil and cannel coal of Tasmania and other parts of Australia will be achieved.

"The tremendous uses made of the internal combustion engine in the conduct of warlike operations in the air, on the land, on the water, and beneath it, give rise to a very heavy demand for light and heavy oils.

METALS.

"Australia to-day produces large quantities of pig iron, steel, and all necesary ferro alloys, and in connection with the latter it may be remarked, in passing that, whilst some use is being made at the present time of the remarkable properties of various ferro alloys (as, for instance, in the wonderful vanadium steel used in the manufacture of the Ford and many other types of motor cars), it is obvious that the scope for the application of these ferro alloys is very great indeed, and that the fringe of the subject so far has only been touched by the technologists of the world.

"The great steel works of the Broken Hill Pty. Co. (originally a leading mining and metallurgical company) and of Hoskins Ltd. are capable of supplying even to date many of the essentials for defence both in iron and steel, and in the innumerable products of the distillation of coal.

"The Newcastle works particularly has already gathered around itself a number of most important subsidiary industries, including galvanizing works, wire-drawing plant, ferro-alloy manufactures, and, with proper encouragement, will doubtless extend in many directions, including such vital industries as tin-plate manufacture.

"Already a capacity of 400,000 tons of basic and foundry pig iron is possible; approximately the same tonnage of basic steel ingots can be made, and therefrom forging blooms and billets, steel rails of all sections, structural steel, beams, channels, and angles, merchant bars of all kinds, and wire rods for wire-drawing, acid steel, shell steel, rifle-barrel steel, ammonia and ammonium sulphate, benzol, toluol, phenol, and naphthalene are being or can be produced.

" A study of the ferro-alloy question in Australia indicates that certain of the vital raw materials are imported, and this is one of the questions which needs to be studied from the defence standpoint.

" All the necessary constituents for brass and bronze are already being produced in ample quantities in Australia, and there only arises the question of the storage of sufficient of these materials as a provision against temporary disablement of any one of the vital plants producing these materials, of which the main ones are copper, tin, and zinc.

" There are also other elements which are necessary, but in smaller quantities, such as nickel and platinum, and the production or provision of such is portion of the whole subject and problem.

MATERIAL FOR EXPLOSIVES.

" Sulphur and sulphuric acid are absolutely necessary, and it is with some pride that I can refer to the work of the staff of the Electrolytic Zinc Co. and the associated companies, such as the Wallaroo, Mount Lyell Co., the Broken Hill Associated Smelters, the Broken Hill South Ltd., the Zinc Corporation Ltd., and others, in the successful development of the difficult metallurgical undertaking for the mechanical roasting of Australian zinc-lead sulphides for the production of sulphur dioxide therefrom in a form completely suitable for the manufacture of sulphuric acid either by the lead chamber or by the contact process. Already thousands of tons annually are being made from Broken Hill concentrates in this way, and early next year a large contact acid plant, producing 60 tons daily of mono-hydrate sulphuric acid, will be in operation at Port Pirie.

" The consumption of aluminium in Australia is steadily growing, and in a short time manufacture from Australian raw materials of metallic aluminium in pig and rolled form will doubtless ensue, and will again provide another link in the chain. The development of the all-metal aeroplane, which will probably be a type of battleplane in the future, suggests the advisability of fostering the aluminium industry.

" Cotton and glycerine are also required in connection with the manufacture of explosives and propellants, and fortunately both these materials are available in ample quantities in Australia already. This year on a conservative estimate Australia has under cotton over 98,000 acres.

CHEMICAL INDUSTRIES IN INFANCY.

"To-day, only the beginnings of the heavy chemical manufacturing trade in Australia and work connected therewith exist, and there must certainly arise a larger electrolytic caustic soda and chlorine industry, which will provide the most essential alkali and at the same time chlorine and hydrochloric acid, which have many uses in the arts, and are also essential for defence.

"The work that has been done to date on the commercial scale in connection with the production of caustic soda and chlorine is due to the Mount Lyell Mining and Railway Company, so that to-day the only commercial plant operating in Australia is at the Yarraville works of this company. This is another instance of the value to Australia of the mining and metallurgical industries.

"Without nitric acid none of the ordinary forms of propellants and explosives can be manufactured, and, whilst considerable quantities of this material in the form of nitrate of soda can be stored for indefinite periods, it would appear as if, during the coming years, plants for the manufacture of nitric acid should be erected in Australia, so as to remove the necessity for importation from across the Pacific.

"The manufacture in Australia of essential medical supplies of all kinds, including anæsthetics, has during the past decade made very considerable progress, but there are doubtless directions in which we are not to-day self-contained.

MOBILIZATION FOR WAR.

"One could traverse practically the whole field of modern industrial production and point out a valuable use for the product in time of war. The fighting services in these days of the nation in arms call to their aid most of the agencies by which the community satisfies its needs in peace time.

"Metals of all kinds find a plenitude of uses in guns and ammunition, in warships, tanks, and aeroplanes, in electrical apparatus, barbed wire, fixed defences, field entrenchments, and billets. In the transportation services machines are replacing animals. Steel is replacing leather.

"Mobilization for war means more and more the mobilization of the resources of the nation's war material. For every man in the line or afloat there are many tons of metal, chemicals, foodstuffs, and clothing. Metal is used to kill the enemy and conserve one's own man power.

" Modern war raises tremendous industrial problems. Half of the nation is in the field ; the other half is mobilized to render every possible aid to the fighting services.

" Thus it appears that in time of peace there should be the closest possible liaison between the naval and military organization, particularly in their departments of supply, and the scientific and manufacturing community.

" Advisory boards should be available to assist the fighting services in their plans for preparedness. The services should say what they want, the manufacturers advise how best they could meet the demand, and the scientists point the next steps to improvement.

" Government should encourage the establishment of industries which, whilst fulfilling definitely useful purposes in time of peace, have a necessary role upon mobilization. Peace-time production methods might be so planned that the change from a peace to a war footing could be rapidly effected.

" Designs and standards might be determined and made known and interest fostered and experience acquired by the placing of small scale orders for peace training purposes, by remission of Customs dues in certain instances, and by bounties where necessary in others.

DEFENCE AND ORGANIZATION.

" Economic considerations usually determine the location of manufacturing plants, but it is just possible that a closer co-ordination between the defence authorities and manufacturers would have better results in the long run. Or if, on the other hand, essential industries must be located in exposed areas, the same co-ordination might lead to improved measures for their defence in an emergency. The Newcastle industrial area is notoriously a case in point.

" In the civil transportation services there is room for tremendous work. The encouragement by the Federal Government of the use of standard types of railway rolling stock, motor lorries, tractors, and horse-drawn vehicles would probably lead to an eventual economy even in peace time. The standardization of vehicles of all descriptions, and for all purposes, seems good business from every point of view.

"The provision of proper communications—railways, roads, telegraphs, telephones, and wireless stations—is a most important matter. Australia is an outpost of empire, and a land of big distances. Reduce the distances by any and every means, and we improve our economic position; we also improve our ability to mobilize our resources in times of stress.

"If Australia is to fulfil her destiny as a portion of the Empire she must develop industrially. The foundations of that development are being laid to-day, and we have the good fortune to be the pioneers blazing the trail. Let the foundations be well and truly laid.

"Whilst one needs to preserve a sense of proportion, and not at all to subordinate the normal process of development to the requirements of times of war, it is none the less evident that industrial development can be ordered in such a fashion that, whilst there will be no loss of peace-time efficiency to serve the needs of the people, the country will not be found wanting in the essentials when it has to fight for its life.

"One needs to keep one's feet on the ground and have regard always to what is practical. A young country would be unwise to commit herself to financial responsibilities for defence which she cannot afford. On the other hand, she would be equally unwise to make no investments for national security. The object all through would appear to be to build with a plan rather than haphazardly—to make present sacrifice for future security.

"British peoples are not usually credited with much capacity to organize big things. They are supposed to muddle through, and usually at the outset of any of our wars we have muddled through. But when the pressure is properly on us we find that British organization rises superior to most obstacles, and the bacon is duly landed. Shall we trust to this method once too often?

"After our wars we pay off our fighting services, reduce naval and military expenditure beyond safety limits, and turn our attention to trade. The story of the years since 1918 is quite in accord with precedent, and the next war will surely find us out unless we develop our imagination and apply the methods of science and the means of production to the problem of preparedness.

KEY INDUSTRIES.

" It will be seen by this rapid survey, which deals with most of the vital necessities for defence, that Australia is particularly well situated to provide practically all materials that are required, and that, provided a thoroughly sound organization is established and extended from time to time, the difficulties of supplying a naval and military and air force with sufficient means of defence are by no means insuperable.

" We have heard from time to time—and, indeed, until we have become rather tired of the phrase—that there are certain 'key' industries. It is rather a pity that these terms after a time become tiresome, because they are the bearers of very significant facts, and I wish to emphasize that, whether we are tired of the phrase or not, the fact remains that there are 'key' industries, and that these industries form the warp and the woof not only of any industrial system of a country, but the very basis upon which any sound system of industrial and defence organization must arise.

" Professor Sir Edgeworth David, one of the most far-seeing and hardest-working citizens of Australia, spoke, on his return from Europe after the Great War, as follows :—

> " 'The public has no idea what we owe to science for our final victory in the Great War, because the British War Office and Admiralty wisely kept as a secret those things which might be used again to defend the Empire. But here in Australia we are differently situated. We are cut off from chemical supplies. It is necessary to establish basic chemical industries in Australia.'

" Sir John Monash, the general commanding the Australian Corps in France, speaking recently in Sydney, drew attention to the serious position arising in Australia in regard to defence matters, and uttered a most serious warning against any further neglect of vital defence problems.

" The chemical and metallurgical industries to which Professor Sir Edgeworth David refers are undoubtedly 'key' industries. I do not claim that they are more essential than many other industries, but we tonight, at a meeting of the Mining and Metallurgical Institute of Australasia, are especially concerned with mining, metallurgical, and chemical industries.

PERSONNEL.

" This is said in order to lead up to the extremely important observation that the personnel responsible for the mining and metallurgical and chemical industries of Australasia are very definitely essential to Australia. The shareholders who supply the money for the financial attack; the directors who supply the policy and the leadership to the shareholders; the men on the management and technical sides, who, in collaboration and with the encouragement of the directors and shareholders, supply the technical and industrial leadership; and the foremen and workmen who, also in collaboration and industrial good-fellowship with all those above mentioned, produce the goods from the raw material, are all essential to the well-being and to the future of Australia's development.

" The inauguration and extension of the mining and metallurgical and chemical industries of Australia supply the necessary personnel which continuously adds to the wealth of Australia, which turns the raw material into finished products in Australia, and which will act as a nucleus for such extensions of organization as may from time to time be required in the development in other industries requisite for the defence of Australia.

" The normal, healthy human being works because he or she finds it the ordinary, normal thing to do. The instinct to work has been implanted in all of us, and without work as a discipline our fibre is sapped and we deteriorate. To work properly we find it necessary to be logically and reasonably contented. I do not mean slavishly or stupidly contented, because without some divine discontent our best work is not done; but if we know the reason for things, if we realize that we are being logically admitted into the council of affairs, then our work gives us joy and the results are good.

" The industries of Australia, both present and prospective, to which I have referred in the preceding notes cannot be successful and helpful to Australia unless there is a basic content amongst the whole of the personnel associated therewith.

" We have at Risdon, following upon the lines developed at Port Pirie and elsewhere, endeavoured to work with all our associates in the zinc industry upon such lines, modified to suit the particular conditions of our industry, as will gradually evolve, over a course of years, a true, clear, and logical co-operation leading to the highest type of Australian efficiency. Such efficiency must always be based upon leadership and understanding, upon sound discipline, based upon appreciation of all circumstances, and upon due recognition of the highest interests of all concerned.

" To the members of this Institute, many of whom will be in the future the leaders and the mentors of many important new industries in Australia, I commend for study the efforts we have made here towards a sound, logical, and happy relation between the whole of the personnel of industry.

" The highest ideal in this world is one of service. Each man and woman should in this young country of ours be able in due course to determine in which direction, having regard to his own well-being and to the well-being of his dependents, he can best serve the State. Having so determined, he should endeavour by all means in his power to devote such portion of his time, ability, and energy as can be spared from the bread-winning for the day to the common well-being of the nation, so that when his time of service is over he can look back with satisfaction and feel that, as far as he was able, he has added to the common funds for advancement.

Our Resources must be Utilized.

" The field is wide—indeed, illimitable. The war material is at hand, and the workers are insufficient in numbers.

" Limitation of numbers in Australia to-day would be a confession of impotence to utilize the resources at our hand. It would be an admission that we have no right to hold this country, and this, I am sure, will never be the attitude of people of our race.

" A paper of this kind must, from its nature, be general, and really only introductory to a series of special papers dealing with sections and sub-sections thereof. I have endeavoured to set out herein a few general principles often discussed, but seldom followed by action, with the hope that members of this Institute will assist as they can in the solution of the problem."

DISCUSSION.

The Chairman (Mr. W. E. Wainwright) said that Mr. Gepp's address was a departure from technical subjects. It was a subject which assumed a national character, and Mr. Gepp had his very hearty personal congratulations on his treatment of it. He believed that there were a great many people who were peaceful because a policeman was handy, and that some of the nations were like those people. The League of Nations had not prevented the settlement of troubles without the use of arms, and it was essential to be prepared for war.

The Administrator (Mr. Justice Ewing) said it was the first time he had appeared in the capacity of the representative of His Majesty the King. When he looked back and reviewed the history of the British nation he thought of little Scotland, with her handful of people, and how it had stood up against all comers, and ultimately crowned her king as the King of England. Rich lands and rich people did not make a nation. The making of a nation depended upon the character of the people. If there was ever a people on God's earth capable of breeding a great nation, it was the Australian people, because of the stock from which they had sprung. Yet Australia could scarcely hope ever to stand alone. In his opinion, if the British Empire broke, in a few years Australia would be gone. The people had to realize their obligations to the British Empire—that without its protecting hand all their modern civilization and culture would disappear like the mists. What was wanted was a hardy and efficient race, such as Moses evolved by walking his people about the desert for 40 years until the unfit had dropped out, when they went and took the Promised Land.

The Premier (Hon. J. A. Lyons) said that, as one who, for the time being, held a responsibility in regard to the finances of the State, he endorsed everything that Mr. Gepp had said in regard to development, and he also agreed with what had been said as to defence. Australia had the conditions that made the right breed of people, and he had no doubts about the future of the country. It was the policy of his Government to develop the resources that Mr. Gepp had referred to—to attract people from overseas to populate the country and make it great. Population was the vital point, and, with the establishment of industries as referred to by Mr. Gepp, the people would come. He wanted to say that they were fortunate indeed in having in Australia—in Tasmania, in particular—such a man as Mr. Gepp. On behalf of the people and the Government he wished to express appreciation of what Mr. Gepp had done and was doing for Tasmania in innumerable ways. It was to be hoped that Mr. Gepp would be able to see the development of Australia on the lines that he, with his great vision, foresaw.

SENATOR J. E. OGDEN said that there could be no doubt as to where he stood in regard to defence matters. Too little thought was being given to the subject. History teemed with instances to show that wars were generally caused by the desire to provide for the people of some country desiring to extend. Australia, with two and a half people to the square mile, was adjacent to a country which had 300 people to the square mile. The menace was obvious. He was glad that Mr. Gepp had drawn attention to the necessity for the development of secondary industries, particularly in relation to mining. He had been struck by Mr. Gepp's statement that the highest ideal was that of service. Everybody should realize that he owed an everlasting duty to the State. The workman should remember that when he drew his wages he was not drawing it from the employer, but from the industry in which he was engaged. He had no doubt about the future of Australia. He wanted to preach the doctrine of work and interest in work rather than over-interest in pleasure. He thought that Mr. Gepp's address should be published throughout Australia. In the Federal Parliament he would impress the necessity for an adequate defence policy. War could not be abolished, and it was his belief that a country that would not fight could not be prosperous. They should ever be prepared to defend themselves.

MR. D. J. O'KEEFE, M.H.R., said it was a great pity that they had to think about and prepare for another war. While they all hoped that there would be no more war, they ought never to forget that the people who guided the destinies of other countries might not be disposed towards peace, and that they would have to be prepared. Such information as that given them by Mr. Gepp should be widely circulated.

Senator J. H. Payne said there was a danger that the people of Australia might become so afflicted with self-esteem that they would commence to think that they were the people of the world. Such papers as that read did a great deal of good. He was at one with Mr. Gepp in that secondary industries were necessary. He went on to refer to the crying need for the repealing of the measure which limited the employment of juveniles, so that many boys, after receiving technical training at the expense of the Government, were unable to get employment. He agreed on the necessity for improved defence measures.

Mr. A. C. Seabrook, M.H.R., endorsed what had been said, and congratulated Mr. Gepp on his address.

Sir Walter Lee referred to the value to Tasmania of the Electrolytic Zinc Co., and expressed appreciation of the work of Mr. Gepp.

Colonel R. E. Snowden, M.H.A., said he believed in the conscription of a nation's resources. Wars were not won entirely on the battlefields. A great deal depended on the enterprise and *morale* of the people. He agreed that unless they were prepared to establish great industries they were not going to develop and be able effectively to defend themselves in war.

Mr. Owen B. Williams said that in the Great War the work of the organizer, the research chemist, and the engineer was predominant. He pointed out that within the ranks of the Institute and the societies associated with it were the best organizing brains, the best research chemists, and the finest engineers in Australia. He urged that this fact should be impressed upon the Government, and suggested that a sub-committee should be appointed by the Institute from amongst its members to arrange them into the various classes, such as research chemistry, engineering (inventions), engineering (investigation), and, finally, organization. If citizen training were again established it should be the aim of members of the Institute to encourage their staffs to take up the work as far as circumstances would allow, and it should be the duty of the Institute to see that its members who had proved their ability, not only in the schools and laboratories, but on the practical side in the field, should receive the recognition from the authorities which was their due. In this way the Institute would be of great service to the nation in the event of another war.

MESSRS. J. H. BUTTERS (general manager of the Hydro-Electric Department) and A. S. KENYON spoke on similar lines, and at the conclusion a vote of thanks to Mr. Gepp was carried by acclamation.

ROCK-DRILL STEEL PROBLEM IN AUSTRALIA, WITH SOME SUGGESTED STEPS TO ITS SOLUTION.

BY G. L. THIRKELL, B.SC.

FOR some time past keen interest has been shown in both America and South Africa in the mining drill-steel problem. This interest has extended to Australian mining fields, and Australian drill users, ever desirous of increasing efficiency, are vigorously attacking the problem.

The writer has recently been engaged in the investigation of the problem on two of the biggest mining fields in Australia, and has formed very definite conclusions concerning the causes of many drill-steel casualties. He puts these views forward in the hope that they may be of interest to both drill manufacturers and drill users.

The subject is here dealt with mainly from the aspect of the properties of the steel and the modifications of these properties when subjected to various manipulations.

It is generally admitted that the drilling machine has greatly increased in power and efficiency during recent years, and the consequence is that the machine is demanding better service from the steel. This is being impressed on the drill users by an increase in steel casualties. This fact has led many mining people to the view that the machine has developed so rapidly that the steel maker has not been able to keep pace, and that consequently the steel is failing by fatigue or so-called crystallization. This view is rather depressing. The writer, on the other hand, is inclined to discountenance fatigue as a serious factor under good normal working conditions where the steel is carefully manufactured and supplied to the user under suitable structural and sound conditions, and has subsequently received kind treatment at the hands of the drill sharpener and the drill user.

It is not reasonable to attribute failures to fatigue until it has definitely been proved that the steel is fundamentally sound and is in the best structural condition to resist the stresses it is called upon to withstand, and, this being so, that all fundamental principles of heat treatment have been carefully observed in the manipulation of the steel on the field. This brings us to the following definite starting-points for inquiry :—

(1) The nature and intensity of the various stresses that the material is called upon to withstand.

(2) The collection of complete and reliable data of a quantitative nature.

(3) The various properties of the steel under consideration when subjected to various degrees and rates of heating.

(4) The most suitable properties required in the various parts of the drill to give the maximum resistance to the several stresses these respective parts are called upon to withstand.

(5) The fundamental methods of obtaining these desired properties with the least expense.

This convinces us at once that before making any general statements we must have a substantial working knowledge of the properties of the steel and the fundamental principles of heat treatment. The rule of thumb method is apparently satisfactory while things are running smoothly, but when trouble occurs it is difficult to find the cause unless the practice is based on sound fundamental principles.

From the experience the writer has had of this work, and from observations carefully made both in the sharpening shops and underground, he very strongly holds the opinion that the bulk of the trouble is to be looked for in insufficient appreciation of the importance of better methods of manipulation.

In the earlier types of machines defects in the material and defects due to faulty treatment were not exposed to the same extent as they now are with the modern machines, which are undoubtedly demanding better performances from the drill steel. The writer firmly believes that the drill-steel user may with all confidence look forward to—

(*a*) a considerable increase in efficiency ;

(*b*) a large reduction of drill-steel casualties ; and

(*c*) an appreciable cutting down of working expenses by carefully investigating the methods of manipulation and modifying them so that they may be based on the correct fundamentals of steel treatment. It is nearly always fatal to depart from these first principles.

The writer is not alone in these views, but is strongly backed up by excellent work recently done in America, and for support would refer to—

(1) The very useful and practical paper by George H. Gillman, "Heat Treatment of Rock-Drill Steel," *Trans-American Institute of Mining and Metallurgical Engineers*, vol. lxvi., 1922.

(2) "Drill Steel: its Forging and Heat Treatment," by H. S. Brainerd, *Compressed Air Magazine*, November, 1922.

(3) "Review of Present Status of Drill-Steel Breakage and Heat Treatment," by Francis B. Foley, Charles Y. Clayton, and Henry S. Burnholz, *Trans-American Inst. of Mining and Metallurgical Engineers*, No. 1206 M, 1923.

The writer is very strongly opposed to the idea of departing from the present recognized standard composition for hollow drill-steel until all its possibilities have been fully exhausted, as he feels that there is a tendency to blame the steel before its best properties have been obtained by correct manipulation. It must be borne in mind that alteration in composition introduces a fresh set of variables and complicates the problem considerably.

Since this question is so definitely dependent on a knowledge of the metallography and treatment of steel for its solution, the writer must briefly introduce the iron iron-carbide diagram for the benefit of those readers who are not familiar with it, and, owing to more space not being available, must limit his remarks to its application to this one type of steel only, although it is very strongly advised that those interested in the subject should gain a good working knowledge of the whole diagram. It is regretted that it is not possible to do more here than mention a few fundamental facts connected with the diagram. The writer has previously written an article on the diagram itself for the use of engineers and people interested in steel treatment. This will be found under the title of "The Effect of Heat Treatment on the Properties of Steel," *Commonwealth Engineer*, October and November, 1921.

Fig. 1 represents the diagram, and shows the constitutional changes which take place in straight carbon steels when they are heated or cooled through various ranges of temperature. For instance, take a steel of any carbon content in the molten condition; on cooling it begins to freeze on reaching a temperature shown along the line

ABC. The temperature of freezing being dependent on the carbon content of the steel, as shown by the diagram, it finishes freezing on reaching a temperature shown along the line ADB. It will be seen that as the carbon content of a steel is increased the freezing temperature is rapidly depressed. This is a point of considerable practical importance, since it shows that a high carbon steel cannot be heated to the same extent for working as a low carbon steel, as burning of the steel occurs if, on heating for working or treating, the incipient point of fusion is reached. For a mining steel of about .8 % carbon content this occurs at about 1200° C., or a lightish-yellow to a white heat.

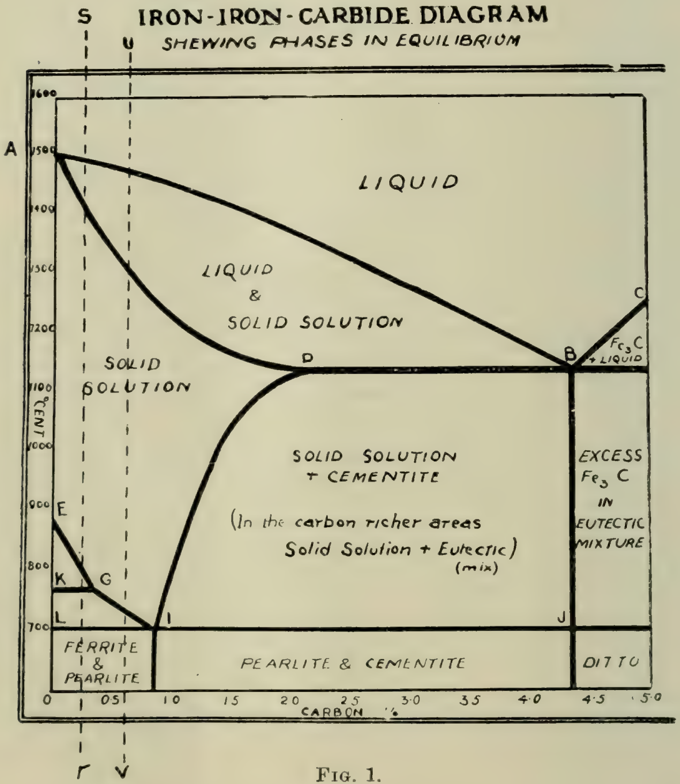


FIG. 1.

Micrograph 1 by 25 diameters (see Fig. 2), unetched, shows a steel burnt in this manner. It will be noted that the crystals are very large, and that the crystal boundaries are oxidized, and there is practically no cohesion between the adjoining crystals. A steel so treated is useless, and cannot be restored. When this happens the burnt portion must be discarded as waste.

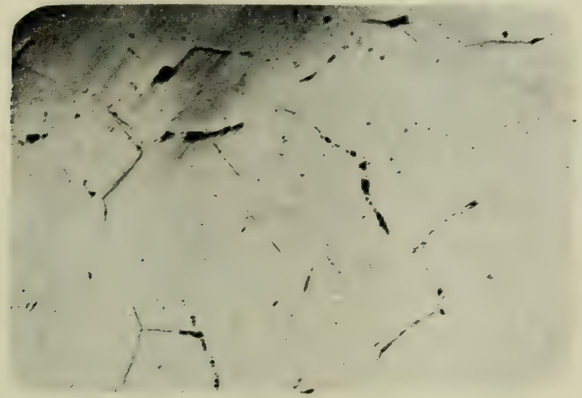


FIG. 2.

MICRO. 1 × 25 diam.—Unetched, showing burnt steel.
By courtesy "*Commonwealth Engineer*" Oct. 1, 1923.

On solidifying, the carbon and iron of the steel remain in a state of solid solution until temperatures along the lines EGID (Fig. 1) are reached. On cooling at the normal rate in air when the temperature falls below the line EGID, iron begins to fall out of the solution in the case of steels below .9 % carbon content, or iron carbide in the case of those steels with a carbon content beyond .9 % carbon; but, on the other hand, if the steel is very rapidly cooled from a temperature above the line EGID, for all practical purposes it remains in the solid solution state. Below the line EGID there are two metallographical constituents in the air-cooled condition as shown in the diagram—viz., ferrite and pearlite in the case of steels below .9 % carbon, and cementite and pearlite in the case of steels above .9 % carbon content, the ferrite being iron, cementite being a carbide of iron, and pearlite being a mixture of iron and iron carbide, the solid solution being a solution of iron and iron carbide.

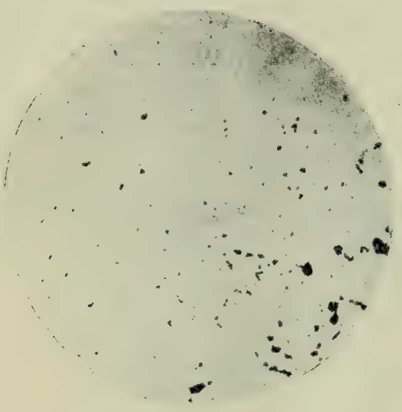


FIG. 3.

MICRO. 2 × 100 diam.—Showing the solid solution state. The black areas show the solid solution in the initial stages of breaking down.

By courtesy "*Commonwealth Engineer*" Oct. 1, 1923.

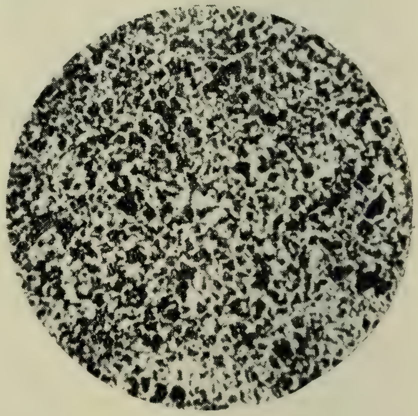


FIG. 4.

MICRO. 3 × 100 diam.—Showing a .4 carbon steel in the normalised or air-cooled condition. The steel structure being of two constituents, pearlite (black) and ferrite (white).

By courtesy "*Commonwealth Engineer*" Oct. 1, 1923.

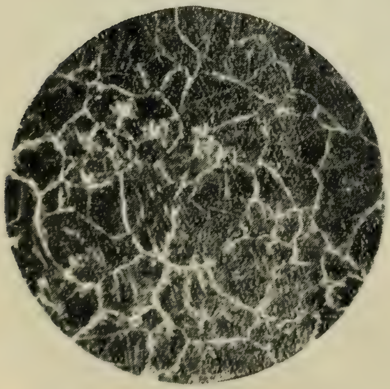


FIG. 5.

MICRO. 4 × 100 diam.—Showing a 1.2% carbon steel in the normalised condition. The black areas are pearlite, while the white areas are hard and brittle cementite.

By courtesy "*Commonwealth Engineer*" Oct. 1, 1923.

On heating up from the cold in the case of the normal air-cooled steel, on reaching the line LIJ the iron in the case of steel below .9 % carbon content, or the iron carbide in the case of the higher carbon steel, begins to pass into the solid solution state, the solid solution state being completed on heating to temperatures above the line EGID; thus, when we wish to harden a drill we heat it to a temperature above the line EGID to put the steel into the solid solution or hard state, and quench it in water to retain the cutting edge in this state. This briefly shows what happens to the constituents.

We next have to consider grain sizes. A coarse grain generally indicates brittleness and a fine grain toughness; therefore the important practical fact in this connection is that, on heating up, the higher the steel is heated beyond the line EGID the coarser the grain will be on cooling; also, soaking for long periods at a temperature above the line EGID coarsens the grain and embrittles the steel. Therefore, the necessity for stressing the importance of temperature control will be appreciated. Further, the higher the temperature from which the steel is quenched for hardening the greater will be the internal stresses set up by sudden contraction, as the higher the temperature the greater the contraction. Some smiths believe that the higher the steel is heated before it is quenched the harder it will become. Study of the diagram will show that this is obviously not the case, as it will be seen that there is a definite temperature at which maximum hardening takes place.

The portion of the diagram that is of most practical importance from the heat treatment point of view is the portion EGIL. This portion is shown on a larger scale in Fig. 6.

On heating up, these constitutional changes take place at some 30° C. higher than they do on cooling—that is, in the case of straight carbon steels. This is indicated by the dotted lines in Fig. 6.

Micrograph 2 by 100 diameters (see Fig. 3), etched in picric acid, shows the steel in the solid solution or hard state. The black patches denote that the hard state is just beginning to break down owing to the quenching action not being sufficiently rapid to retain it entirely in the solid solution state. This is what is generally encountered in actual practice.

Micrograph 3 by 100 diameters (see Fig. 4) shows a steel of .4 % carbon content in the air-cooled condition. It will here be noticed that the solid solution state has broken down into two constituents —the pearlite and ferrite, as before mentioned—the white areas being iron or ferrite and the black areas being a mixture of iron and iron carbide or pearlite; and micrograph 4 by 100 diameters (see Fig. 5)

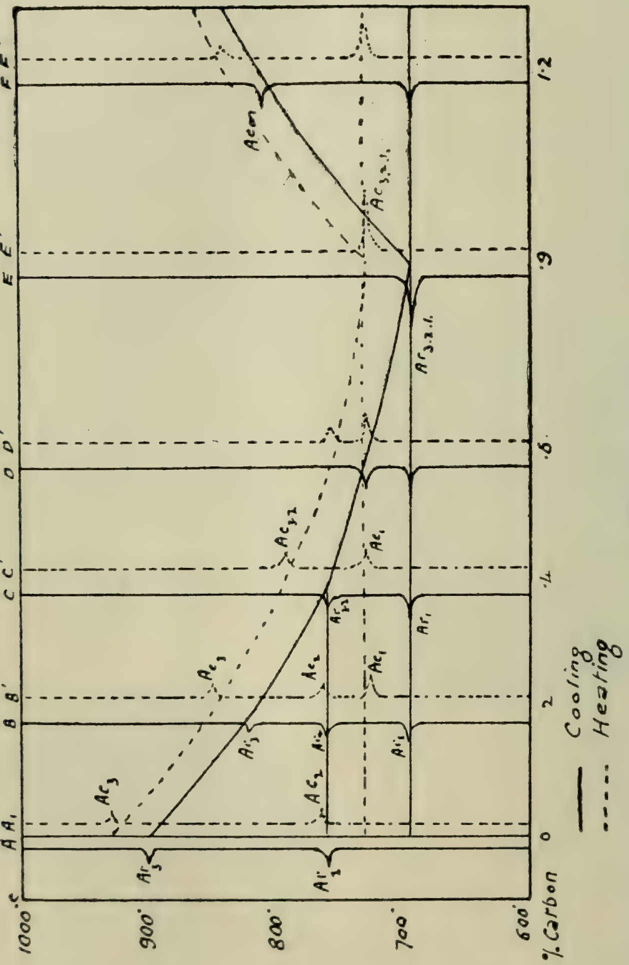


FIG. 6.

SHOWING CRITICAL RANGE AREA EGIL. (Fig. 1. on larger scale).

By courtesy "*Commonwealth Engineer*" Oct. 1, 1923.

shows a 1.2 % carbon steel in the air-cooled condition. In this case we have the solid solution state broken down again into two constituents, the black area of pearlite and the hard, brittle, white crystal boundaries of cementite or iron carbide. Mining steel, in its normal or air-cooled condition, consists almost entirely of the black constituent—pearlite.

It is desirable that the carbon content of a mining steel be kept below .9 %, as the free cementite boundaries which exist in steels of higher carbon content are very hard and brittle, and need special complicated treatment for the modification of this constituent.

For mining steels the writer recommends a carbon content from .7 to .8 %, with a low manganese content, say not higher than .3 %. The importance of considering the carbon and manganese contents together is not sufficiently appreciated. It is not generally realized that one modifies the other to quite an appreciable extent. Thus, if one wishes a high carbon content in mining steel, it is well to keep the manganese figure low, whereas with lower carbon content a little more manganese is allowable and perhaps necessary. The effect of manganese is to increase the depth of hardness, as it retards the breaking-down of the solid solution state.

Referring again to grain sizes, it is stated that a fine grain denotes comparative toughness. It must be explained here that a very fine grain does not always denote toughness, because it may indicate a hardened state, which may be comparatively brittle ; but the writer wishes to point out that in the air-cooled state, in which the steel is comparatively soft and has two metallographical constituents, there is a minimum grain size, even as there is in the hardened state, where there is, from a practical view-point, only one metallographical constituent—viz., solid solution. This question of grain size is of considerable importance in both cases, as it represents the toughest condition for either state, and hence is the most resistant structure to withstand shock. Hence those portions of the drill that have to withstand shock, compression, bending, and torsional stresses should be in the soft condition, but have the finest possible grain for that condition, while those portions of the drill that have to do the cutting and resist wear will have to be in the hard state. but, again, must have the finest possible grain obtainable for that state. When this particular point is appreciated, and the method of obtaining these conditions fully understood and practised, very great progress has been made in the solution of the problem.

Limiting our attention to steel below .9 % carbon content, and referring again to the diagram Fig. 1, the area EGIL is known as the critical range, and from now on will be spoken of as such. It has been mentioned that the grain size increases as the temperature of the steel is increased above this range, or, again, with the time the steel is allowed to soak at any temperature above it. It is said that forging or working refines the grain. This is so at temperatures above the critical range, and if working is finished at a temperature just above this range a relatively fine structure results ; but if working is carried on below the range the grains are merely distorted, hardened, and strained, and the toughness of the steel is impaired. This is called cold working. If this cold working be too severe microscopic flaws or clinks result, and subsequently lead to mysterious fractures and failures. This is another question for very close attention. If not too severely cold-worked these strains may be relieved by re-heating to a temperature a little above the critical range, holding at this temperature for sufficient time for the crystals to rearrange themselves, and then allowing the steel to cool in air. This is called normalizing, and the process is used to refine the grain in addition to removing forging stresses. The finest grain in the case of the hardened steel is obtained by quenching from a temperature just above the critical range—*i.e.*, at a medium red heat on the dullish side in the case of mining steels.

The next question of vital importance is the rate of heating, apart from the question of degrees of heating. This is where many mistakes are being made on mining fields. It is a matter of relative expansion and toughness of material, and, when looked at in this light, it is easy to imagine what may happen when hardened drills, such as blunts returned to the shop for re-dressing, are rapidly heated for forging. The cold, hardened under layers of steel are severely stressed by the expansion of the rapidly-heated outer surface, and frequently cannot withstand the strain, with the result that small microscopic flaws are produced, and again lead to mysterious failures. It must be remembered that hardness is obtained at the expense of toughness, and obtaining the best properties of the steel for any specific purpose is a matter of compromise.

THE VARIOUS PROBLEMS UNDER CONSIDERATION AND THE SIMPLE APPLICATION OF FUNDAMENTAL PRINCIPLES OF STEEL TREATMENT AND MANIPULATION.

What has been said in the earlier portion of this article may be taken as a necessary introduction. It is now intended to deal with the specific problems which have presented themselves on the various Australian fields visited.

While investigating this question the writer was given every facility and much valuable assistance in making necessary observations, both in the tool shops where the drills are prepared and underground where they are used, and, basing the conclusions arrived at on these various observations, now submits his views regarding the possible and probable causes of failures under consideration, and gives suggestions for reducing the present percentage of failures and for increasing the boring efficiency of the drills themselves. The several problems under consideration are—

(1) Fracture of the drill along its length—*i.e.*, in those parts which have not been subjected to any form of heating or treating liable to modify the structure of the steel as it is received on the field.

(2) Fracture of the drill in the vicinity of the bit—*i.e.*, in those parts of the steel where the original (as received) structure has been modified by that treatment locally.

(3) Fracture of the drill in the neighbourhood of the shank—*i.e.*, also in a part of the drill where the original (as received) structure of the steel has been modified by local treatment.

(4) The boring life of the bit subsequent to each re-dressing.

(5) The most economic method of manipulation to give the desired results.

(*a*) GENERAL CONCLUSIONS.

It appears to be thought by many users in Australia and in America that the development of the drilling machine has overrun the development of the steel, but, from the work that has been done on the subject, and from observations both underground and in the sharpening shops, the writer is inclined to discountenance fatigue as a serious factor under normal working conditions where the steel is reasonably sound and in a suitable structural condition, and where accidental causes are not a factor. Experience on the fields tends to confirm the writer's previous conviction that much of the difficulty is to be looked for in the primary selection of the material and its subsequent manipulation on the field.

Since there are many small factors, which, unfortunately, are not very obvious, which cause failures in all types of tool work, and since the facilities for getting through the quantity of material required on mining fields are not sufficient to eliminate the personal element of the men manipulating the material, it is quite reasonable to expect that some of these points might be missed and lead to a percentage of unsatisfactory tools. This is a thing that is definitely occurring. The writer has been fortunate in finding samples of practically every type of failure and defect, to be mentioned later, while observing the normal working practice on some of the fields. It is therefore reasonable to suppose that by illustrating these defects, giving reasons for the various failures that result, and simple suggestions of how to avoid each in turn, some considerable improvements in the performance of the steel may result.

It is the writer's opinion that the cutting efficiency of the bits is even a more important matter than the actual question of casualties, although the two, as will be seen later, are very intimately related. The suggestions to be put forward will be based on fundamental principles, and can in no way be harmful, but, on the contrary, will tend to improve results considerably, since the bulk of the defects noticed were very definitely due to the neglect of some fundamental principle.

The first essential move towards greater efficiency is the standardization of treatment based on these underlying principles, and this should be followed by definite research on gauging. The gauging is placed second, because if the treatment of the steel is not uniform the gauging data must of necessity be unreliable. At the same time, the gauging question is one of the utmost importance. The writer is of opinion that—

1. The main causes of these troubles are—
    - (*a*) A certain amount of abuse of the steel by the user.
    - (*b*) A certain amount of unreliable gauging, owing to the non-uniform condition of the bits in use.

(*c*) Lack of appreciation of essential fundamentals on the part of the sharpeners. (This is only to be expected, since it is quite obvious that these men cannot be in possession of all the modern research data on steels.)

(*d*) Lack of careful inspection of drills before they go underground.

(*e*) Incorrect methods of heating for forging and hardening.

(*f*) Lack of attention to the condition of forging, tools, etc.

(*g*) Insufficient heating capacity for the amount of drills required to be got through in a definite period. This leads to a tendency to rush the work, which in many cases gives unfortunate results.

(*h*) Use of wrong quenching media.

(*i*) Insufficient touch and exchange of ideas between the drilling-machine manufacturer, the steel maker, and the user on the field; also insufficient touch and exchange of opinions between the drill sharpener and the drill user.

(*j*) Forging and hardening in one heat, which is absolutely unreliable, in addition to increasing the internal stresses in the hardened steel.

2. The source of supply of materials is of the utmost importance, since the bars of steel have to be brought down to the required size from ingots. The steel has to go through many operations at the works where it is produced, and these fundamentals apply just as much in these works as they do in the sharpening shops on the field. Therefore, in selecting the source of supply of drill-steel bars, it is advisable to select a manufacturer who uses the most modern and up-to-date methods of production based on extensive scientific research. Defects liable to occur in these bars will be dealt with under "Failures and Causes."

3. At the present time it is quite undesirable to change the composition of the steel, since in many cases where troubles are arising the best results are not being got from the particular type of steel recommended as a standard for this type of work. It might be noted here that in America, owing to the opinion being in vogue that the machine had jumped ahead of the steel, and hence fatigue being considered a main factor accounting for failures, alloys were introduced, but the introduction of alloys into steel increases the sensitive-

ness of the steel to treatment and entails higher initial cost, and is apt to lead to lack of reliability unless very skilfully treated. As far as the writer knows, little progress has so far been made by the adoption of alloy steels, and the so-called straight carbon steel of from .7 to .8 % carbon content, and manganese content in the neighbourhood of .3 %, is to-day recognized as standard material for rock-drill bits. This material is about the simplest there is to treat, and allows a wider latitude in manipulation than either alloy steels or steels of higher carbon content. Further, it has an advantage of saving time and expense, owing to the fact that it can be used in the straight-out hardening condition without needing any subsequent tempering treatment, such as is required by higher carbon steels and most alloy steels. It is one of the best abrasive steels for this type of work. With careful treatment a steel of this nature gives a very hard cutting face with a softer and tougher structure supporting it underneath, whereas an alloy steel would tend to harden throughout. As previously mentioned, hardness is generally obtained in steel at the expense of toughness. This brings us to the consideration of the stresses that the drill is called upon to withstand. These are mainly shock, torsion, abrasion, compression, and bending. The obvious properties required, then, are toughness, abrasive hardness, and shock hardness.

4. Altering composition introduces further variables and complicates the problem enormously; therefore, the better way is first to look for the causes of failures that have been experienced and to try to eliminate them by suitable manipulation. Change of composition should only be considered when it is found impossible to get any further after the possibilities of the best methods of manipulation based on sound fundamentals have been exhausted. The writer feels quite confident that this has not yet been done, and it looks as though a large improvement will result from closer attention to the first principles of treatment and manipulation securing uniformity in the drills, and then closely studying the question of gauging.

5. The writer does not advise trying to adopt different tempers for different ground, but strongly advocates that only one standard temper be adopted, and that temper should be the best possible temper obtainable for the severest conditions of service. Obviously, if the drills will stand up to these conditions they will do very much

better in easier ground than would be the case if softer bits were used.

6. The drill smiths should be encouraged to read suitable literature on steel properties and manipulation.

7. As the methods of dealing with the steel question on the mining field generally have to be considered from the economy standpoint, and since large quantities have to be dealt with in a comparatively short space of time, it is questionable whether it is desirable to instal refined scientific methods of control requiring highly-paid technical men for supervision, especially as practically only one type of steel has to be dealt with, and the type of steel recommended is one which is fairly easy to treat by a man of good average intelligence who is interested in the work, observant, and careful, and, if he checks his judgment occasionally as to temperature by, say, Metcalf's test (which will be described later), carefully examines the blunts returned to the shop for re-dressing, compares notes with the men operating the machines, and correlates the various observations with the fundamentals of treatment, sufficiently good results may be obtained. The writer suggests that these men should be given every facility and encouragement to help their judgment, since the efficiency of the tools lies in their hands.

Several very important points are :—

(*a*) Furnaces in which the steel bits, shanks, etc., are always under the man's direct observation while heating, and in which he has a sufficient heating capacity for the amount of steels he is required to put through in a given time—*i.e.*, for carrying out the work on the lines to be suggested.

(*b*) A sufficient quantity of efficient quenching media.

(*c*) Uniform lighting of the shop in the vicinity of the furnaces and quenching baths. This light should be, as far as possible, always the same, and of a dullish nature, in order that the judgment of colour may be uniform.

(*d*) Ventilation of the shop is of importance, and is best effected from the roof if possible. This is mentioned because the smith is apt to imagine that working with a small furnace is hotter than working with a blacksmith's forge fire, and so is prejudiced in favour of the latter, which is a very primitive and a much more unreliable method.

(b) FAILURES AND DEFECTS AND THEIR CAUSES.

1. Steel defects in the bars as received from the manufacturer—

(a) Laps caused by folding over the metal during the forging or rolling operations.

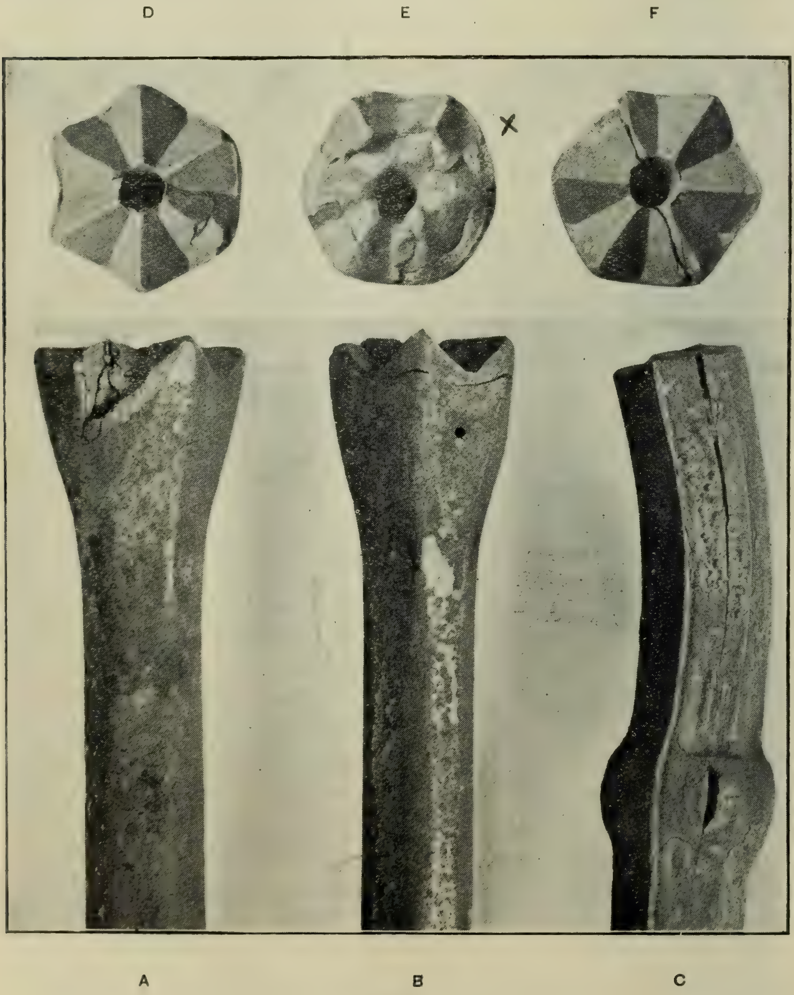


FIG. 7.

(*b*) Seams which may be caused by laps being drawn out along the length of the bar during rolling, or may originate from the cold pouring of the ingot.

(*c*) Roaks—seams running along the length of the bar, generally caused by the inclusion of impurities.

(The results of (*b*) and (*c*) may be clearly seen in Figs. 7*c* and 7*f*.)

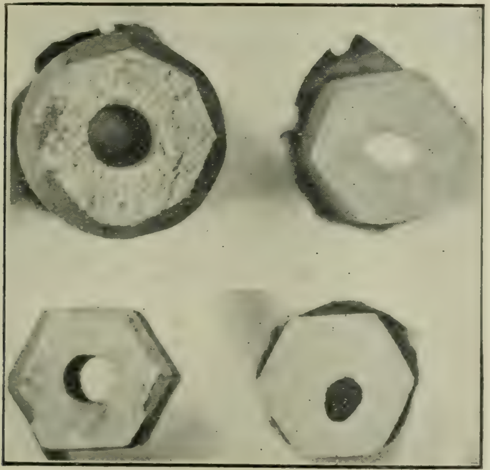


Fig. 8.

Showing good centre-holes and bad centre-holes.

(*d*) Clinks or small cracks caused by over-stressing the metal during working and causing microscopic fractures; these may be either circumferential or longitudinal.

(*e*) Unsuitable structure—*i.e.*, the structure of the steel not suitably modified for the required conditions of service.

(*f*) Excessive impurities, such as sulphur and phosphorus, causing undue brittleness.

(*g*) Slag inclusions, causing small hair lines and starting-points for the propagation of defects.

(*h*) Faulty centre-holes through the bars. Fig. 8*a* shows bad centre-holes in sand-cored bars, uneven, out of centre, and even sand-filled, while Fig. 8*b* shows good holes manufactured by a good maker who is no longer using the sand-core process for the production of the steel. The inside walls of the steel produced by the sand core method are rough and pitted, and possess many small indents which may occasionally lead to a defect ; while the inside walls of the sample shown in 8*b* are smooth and regular and free from defects. Frequently many mysterious failures may be explained by these defects existing in the bars in the original (as received) condition. In a recent investigation on a big Australian gold-mining field, where there was an unduly high percentage of breaks along the length of the drills, the writer found circumferential clinks in the stock bars before they were put into service. These clinks will certainly account for a fair proportion of these breakages, if not for the majority. The bars should be carefully inspected before being cut up into lengths for drills. In many modern steel works, where the manufacturing practice is of a high standard, there is an efficient inspection department, the function of which is to cull out defective material and prevent it leaving the works.

2. Defects initiated or developed in the sharpening shop and by accidents or abuse underground—

(*a*) Clinks caused by cutting the bars with a cold set and fracturing instead of cutting through with a hack saw machine. This practice starts small microscopic cracks, which may be propagated on heating for forging, during the forging operation, on re-heating for hardening, and on quenching for hardening. These may not be noticeable to the naked eye for a time, but on successive re-dressings of the bit may be considerably extended so as to cause a splitting-off of the wings, or even the fracture of a bit down the centre. It is also likely to cause fracture from the centre hole outwards. This particular vice is one of the causes contributing to such defects as are illustrated in Figs. 7*a*, *d*, and *f*, which show samples of popper-drills collected by the writer from an ordinary routine batch which was being treated by the ordinary practice employed on one of the fields visited.

(*b*) Rapid heating through the initial stages for forging, hardening, etc., causing cracks due to unequal and sudden expansion. This is an exceptionally dangerous practice when heating blunts for re-dressing, it being perfectly obvious that the blunts are already hard, and hence have little ductility. On being suddenly heated they are asked to expand considerably before the temper of the steel is sufficiently drawn to allow of sufficient toughness or ductility to permit the forced expansion. Small cracks, or even large cracks, causing fracture or even ringing-off of the bit, might often result from this cause. This is another very important factor contributing to defects shown in Figs. 7*a*, *b*, *d*, *e*, and *f*. Fig. 7*e* is of special interest in this respect; if the fracture is carefully observed it will be seen that the fractured area in the vicinity of the hole shows a rubbed and scaly appearance, which clearly shows that this area was in an internally fractured condition during the forging operation, proving that the damage was done either in the heating for forging or during some previous operation. Fig. 7*a* also shows the walls of the vertical crack in a scaly and rubbed condition, proving that the defect occurred prior to the last hardening operation. The condition of the bit shown in Fig. 7*b* may also be due to the practice of rapid heating or to some carelessness in a previous hardening operation, or perhaps a little of each. Obviously none of these bits are suitable for the work they are required to do. The majority of the batch from which these samples were selected showed defects, and, needless to say, the smith wanted to blame his steel.

(*c*) Quenching at too high a temperature, causing unnecessary additional contractional stresses in the metal, and at the same time giving a coarse cyrstalline structure of intense brittleness. At the point X in Fig. 7*e*, just a small area where a wing was trying to hold on to the bit, the structure will be seen to be very coarse, indicating an unduly high hardening temperature, in addition to the other unkindnesses to which this wretched batch was subjected.

(*d*) Cold working—*i.e.*, the forging or working of the steel at temperatures below the critical range or hardening temperature. For tnis class of steel this range occurs at about a medium red colour. This practice strains the metal unduly, and is apt to cause microscopic cracks on the outside or fracture on the inside, which is not obvious to the naked eye. In addition to being a possible cause of fracture of bits, this is a very important matter when dealing with shanks. Figs. 9*a* and *b* show two broken shanks from drills which failed while the writer was visiting a field recently. Fortunately, the fractures were clean and new. At the point X in Fig. 9*a* a distinct flaw can be seen, and, since this flaw is oxidized, it is obvious tbat it existed before the shank was hardened, and certainly before the drill went into operation. The most probable cause of failure is a "clink" due to cold working. Fig. 9*b* is even more interesting. This time the defect starts from the centre hole and works outwards. Evidence also exists here that the defect was there before the drill shank was hardened or put into operation, and it was only reasonable to expect the drill to fail.

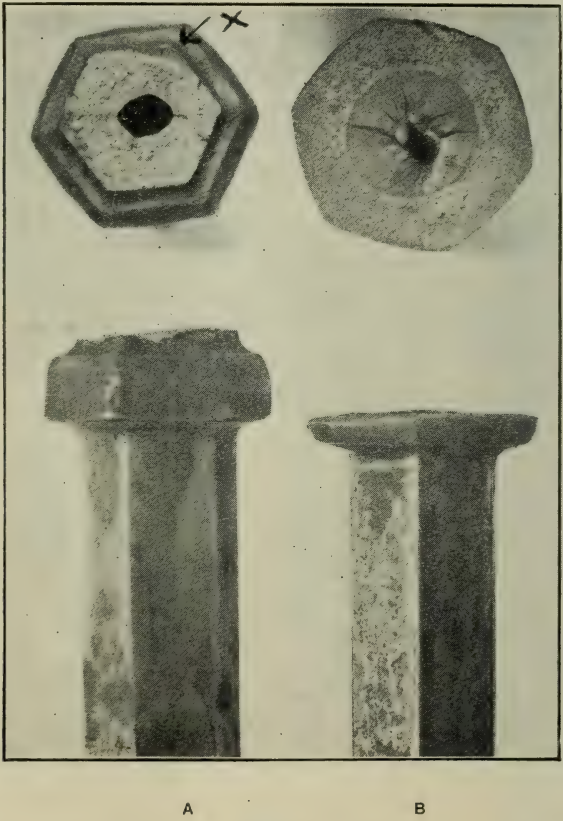


Fig. 9.

If the reader will now refer to Fig. 10*b*, at the point X a channel will be noticed around the inside of the collar, caused by the jumping up. If the collar is worked too cold the forging stresses will tend to concentrate at the bottom of the channel, and in time tend to produce flaws. The writer is of opinion that the failure shown in Fig. 5*b* was primarily due to cold working the collar.

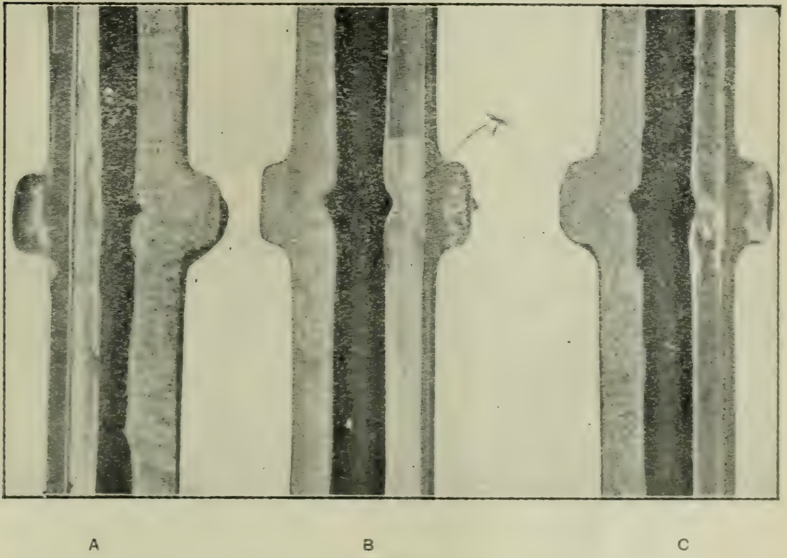


Fig. 10.

Showing structural modifications due to various manipulations.
Brainerd Comp. Air Mag. By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

(*e*) Laps caused by (1) careless forging methods folding in the metal, scale, etc. ; (2) bad tools with unduly sharp corners, etc.

(*f*) Burnt Steel.—This is caused by heating the metal too high and reaching the incipient stage of fusion, which causes the crystal boundaries to become oxidized, as shown in Micrograph 1 (Fig. 2), and hence breaks down the cohesion between adjacent crystals Material of this nature cannot be restored, and any parts accidentally so treated should be cut clear from the sound material and discarded.

(*g*) Overheating, causing cracks, brittle structures, and decarbonized bits and reaming edges. Some smiths believe that the higher they heat the steel the more efficient will be the hardening effect. This opinion is entirely wrong, since hardening occurs at a definite point, and, if the steel is efficiently quenched at a temperature just above this point, the hardest possible structure with the best wearing properties is obtained. Beyond this point the crystals increase in size both with rise of temperature and length of time held in the range, and the coarser the structure the more brittle the steel, and it is a fact that when quenched from a very high temperature a softening effect is produced on the steel.

This may be partly due to the hot centre drawing the temper of the hardened outside of the steel. The effect of overheating can be modified by re-treatment, but in the case of burnt steel re-treatment is useless.

(*h*) Sharp corners, causing local concentration of stresses. These must be avoided wherever possible. Fillets and rounded shoulders should replace them wherever permissible. As stated by Dr. Andrew M'Cance in his paper on "Stress Lines in Steel After Permanent Deformation" (*Trans. Inst. Engineers and Shipbuilders in Scotland*, vol. lxii., part 6)—"If a sharp corner occurs at any change of section under stress it will lead to an intense localization of high stress and increase to a very great degree the chances of forming cracks through fatigue."

A further important and interesting reference to the effect of notches and discontinuity of surface is made by Dr. Andrew McCance in his contribution to Hatfield and Thirkell's paper on "Season Cracking" (*Journal of the Inst. of Metals* No. 2, 1919, Vol. XXII., page 113).

This applies particularly to the inside of the shank collar, as shown in Fig. 10*b*, and to the corners at the collar on the outside of the shank. Sharp corners on forging tools may mark the steel in places and cause the concentration of stresses either while the steel is being worked and heated or while the steel is in actual operation. The formation of these sharp indents and corners is particularly dangerous while the steel is being worked in what is known as the blue brittleness zone—*i.e.*, at a temperature just below a black-red heat. This point is of peculiar interest, since on two of the mines it was noticed that in very many cases the position of the fracture along the length of the drills corresponded with the outer edges of the gripping vices in the sharpening machines. The outer edges of these vices in some cases are quite sharp. In long drills it was noticed that a very strong vibration was set up by the forging operation. The temperature of the steel at this point would most likely lie within the blue brittleness range—*i.e.*, in this range the steel would be in a condition in which it would be most sensitive to shock and vibration caused by the forging operation. The effect of this vibration on a long length would be to concentrate the stresses at the junction of the steel and the outer edge of the vice. The writer would strongly advise that wherever possible a rounded shoulder be put on the outside edges of these tools.

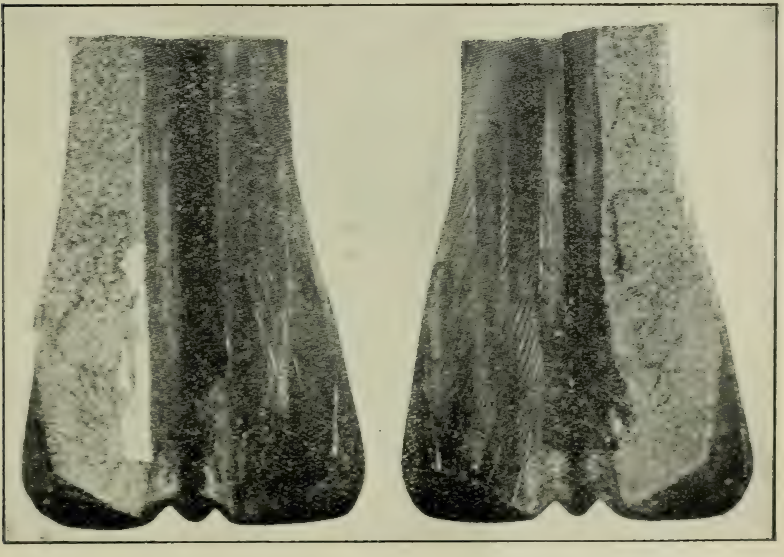


Fig. 11.

Showing forged structure behind hardened bit.

Brainerd Comp. Air Mag. By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

Speaking of blue brittleness, it is interesting to note that a mild structural steel at atmospheric temperature has a maximum stress of approximately 27 tons per sq. in., with an elongation per cent. of about 38, whereas a test piece of the same material being pulled at between 300 and 400° C. has a maximum stress of about 32 tons per sq. in., with about 23 % elongation. Here it will be noticed that the ductility of the material has fallen off considerably. It will also be noticed, on reference to Mr. Harry Brearley's book on the heat treatment of tool steel (page 56), that the breaking force required to break a piece of warm tool steel drops from about 42 units at approximately 150° C. to about 22 units at approximately 450° C., and then rapidly rises again to somewhere about 50 units, showing that at a black heat the steel is very much more sensitive to shock than it is when cold or at a higher temperature.

(*i*) Sudden change of structure. This is what may be expected to occur in the case of a forged bit cooling down from a high temperature and being re-heated for a very short length from the bit face and quenched to a fixed depth of, say, a half inch only in cold water. This leaves a coarse, forge-stained structure behind the bit, with a fine structure in the chilled half-inch length and a very sharp dividing line, which is apt to lead to cracking or ringing off of the bit. Fig. 11 shows the steel treated in this manner. The coarse forging structure may be clearly seen behind the hardened structure. Quite definitely the

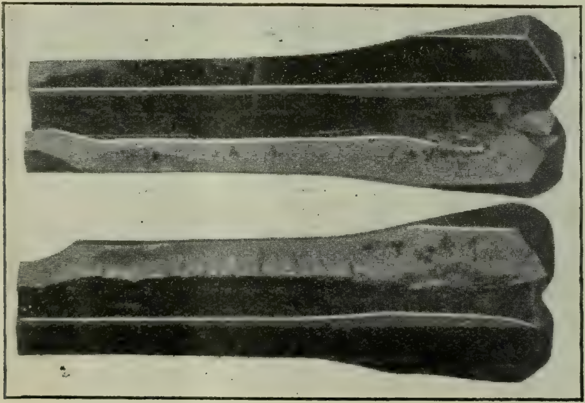


FIG. 12.

SHOWING THE EFFECT OF RE-HEATING THE BIT BEYOND THE FORGED PORTION TO REFINE THE GRAIN AND REMOVE THE FORGING STRESSES PRIOR TO HARDENING, THE BIT BEING HARDENED FOR $\frac{1}{2}$ TO $\frac{3}{4}$ OF AN INCH BEHIND THE FACE ONLY.

Brainerd Comp. Air Mag. By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

best practice is to re-heat the forged portion to a temperature just above the critical range and let it soak for some 10 to 15 minutes to allow the structure to rearrange itself, and then to let it cool in air to a black-red heat, or even cool right out, and then re-heat the bit to the hardening temperature for a distance of about three-quarters of an inch behind the bit face and quench right out. The effect of such treatment will be seen in Fig. 12. The forging strains are removed, the structure is refined, and the drills will be uniform in treatment. By method shown in Fig. 11 the treatment cannot be uniform, but if the forging is finished at a temperature round about the hardening temperature quite good results may be obtained without re-heating or normalizing, and the writer feels that many mining people will prefer to use this treatment rather than adopt the extra refining recommended; but it must be borne in mind that extra risks are taken by not adopting the refining treatment.

The writer strongly recommends normalizing in the case of shanks. Fig. 10*a* shows a shank as forged with the end subsequently hardened, while Fig. 10*b* shows the shank refined beyond the portion heated for forging, and Fig. 10*c* shows the shank in the as-forged condition only. The writer recommends refining the shank so as to remove the forging strains and put the steel into the best structural condition to withstand the stresses to be encountered in service.

Again, the formation of a sudden hardening line, such as may be caused by quenching to, say, a half-inch only and allowing the drill to rest on a submerged plate, is dangerous, and may lead up to the ringing off of bits. The change of structure from tough pearlitic or sorbitic to hard martinsitic should be gradual, and this may be effected by carefully watching the method of quenching.

(*j*) Undue internal stress induced by forging and hardening in one heat—*i.e.*, the forging operation induces a certain amount of internal stress, and the quenching operation, causing sudden contraction, adds considerably to this inherent stress. The writer definitely and absolutely condemns the method. The results of this method may be seen in samples shown in Figs. 7*a*, *b*, *d*, *e*, and *f*, and in Fig. 13*a*, which is of special interest, since it is a sample taken from the normal practice of one of the fields visited.

The sharpening smith in this case was asked to harden 5 drills by his usual practice. These were taken underground by the writer and the underground manager of that particular mine, so that their performances might be observed. Fig. 13*a* shows what happened to three of these drills. Even this is sufficient to show the danger of the practice of forging and hardening in one heat.

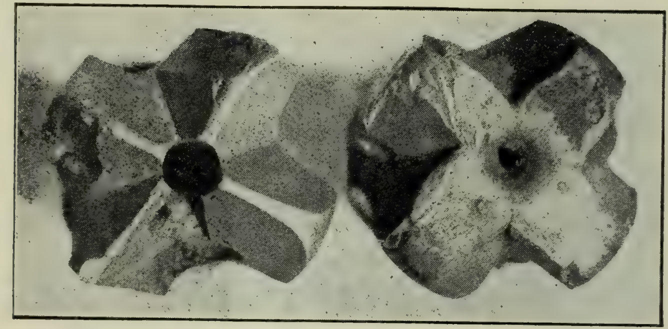


FIG. 13.

SHOWING FAULTY TREATMENT OF BITS—
A. FORGED AND HARDENED IN ONE HEAT. B. INEFFECTIVE HARDENING.

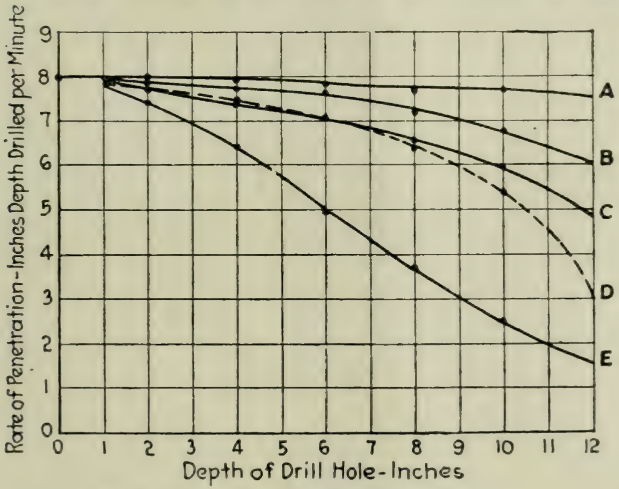
(*k*) 1. Faulty bits—hard bits quenched from too high a temperature causing chipping and loss of wings. This leads to undue brittleness, loss of gauge, and extra duty on both the steel and the machine, and extra depreciation.

2. Soft bits, caused by not raising the heat sufficiently high prior to quenching for hardening, not allowing sufficient time for the heat to become uniform along the cutting edges of the bit face, or allowing the temper to be drawn too much by not quenching the bit efficiently. These lead to loss of boring speed, unduly blunt drills, loss of gauge, and extra duty on machine and steel.

3. Soft centres, caused by insufficient time being allowed for the bit faces to become uniformly heated—*i.e.*, the tips of the wings become heated first, and the heat travels towards the centre hole of the drill. This can generally be detected by glancing at the drill face before quenching. If the heat is not uniform it will be noticed that the colour in the centre of the drill is a shade duller than that on

the outside. This particular defect may also be caused by a collection of steam around the hole, preventing the water coming in contact with the steel in quick enough time to effect hardening. This is liable to be caused by placing the hot drill direct on a tray with only about half an inch or so of water over it. The effect will be clearly seen in Fig. 13*b*.

4. Lack of uniformity of hardness across the bit faces. This is again due to uneven heating, and is liable to cause an abnormal loss of gauge and undue bluntness. The effect of these various bit defects is illustrated in Figs. 14 and 15, which include a graph and a few illustrations taken from



Rock, Barre Granite, Initial diam. of Bit 1⅞ in.

FIG. 14.

GRAPH ILLUSTRATING THE EFFECT OF MANIPULATION OF HEAT-TREATMENT ON THE CUTTING EFFICIENCY OF THE DRILL BIT. AFTER DRILLING A DEPTH OF 12 IN. IN BARRE GRANITE.

George Gilman's paper on the "Heat Treatment of Drill Steel," which paper the writer most heartily agrees with, except that he does not altogether advocate the use of cyanide in hardening shanks at the present juncture, as the process is risky unless carried out by an experienced man.

It is useful to note here, while discussing bit defects, that stress cannot be too strongly laid on the need for consistent lighting of a dull nature in the hardening and forging shops, and also on the constant checking of hardening temperatures by the fracture method to be given under "Manipulation."

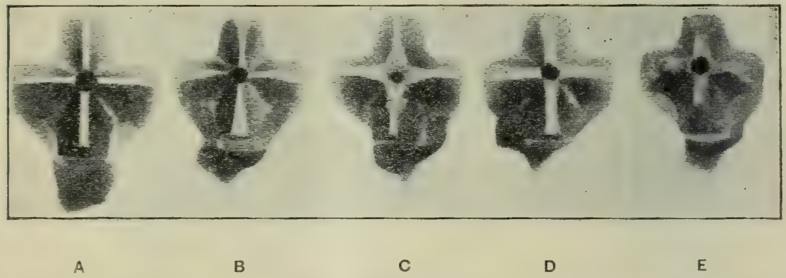


FIG. 15.

A. Forged between reasonable limits of heat, say from orange on the top side and finished at a medium red, i.e., just above the hardening temperature, allowed to cool down and re-heated to a medium red on dullish side and water quenched. This bit shows good treatment, giving a good cutting edge and minimum gauge loss, about 1/64 in. after boring 12 in.

B. Was forged at a much higher temperature, dangerously near the burning zone, cooled down from a higher temperature than A. Then re-heated to the same temperature as A. and water quenched. The boring efficiency is impaired and the tips of the wings unduly worn.

C. Soft centre. The bit was forged under the same conditions as A., allowed to cool and re-heated rapidly at a hotter temperature until it reached a medium red colour similar to that at which A. was quenched, the result was that there was insufficient time allowed for the centre of the bit to reach the hardening temperature prior to being quenched. Thus only the tips of the wings which would heat up first have become hardened. A similar result to this is liable to happen if an undue amount of steam is permitted to form round the hole and thus keep the water there from contact with the steel.

D. Forged in the same manner as A., cooled, then re-heated to a lower temperature than A. prior to quenching. Bit is unduly blunt owing to not having been heated sufficiently high for hardening correctly. A soft bit impairing the rate of boring.

E. Forged correctly as in the case of A. above, cooled and re-heated to a high temperature prior to quenching for hardening, this caused an unduly brittle coarse structure resulting in chipping off of the wings and chipping of the cutting edges, thus considerably reducing the boring speed and the gauge. This is the sort of thing that is likely to result from forging and hardening in the one heat; loss 1/16 in. gauge.

Another type of defect that may occur with bits is that the skin only may be hardened. This may be due to inefficient heating and quenching—*i.e.*, if a fair amount of heat is left in the bit, and the bit is only quenched for about half an inch of its length, the temper will be allowed to draw too far towards the outside, causing only a thin, hardened skin, which will very soon either wear off or chip off in use, exposing the softer material underneath, which will lead to undue wear of gauge and undue blunting of the bit. The result of this kind of treatment will also be similar to that shown in Fig. 13*b*. On the other hand, the bits may be hardened too far back, causing brittleness and a tendency to ring off. This may be prevented in either of two ways, preferably by not heating the bit too far back when heating for hardening, or by careful manipulation in the quenching operation—*i.e.*, by varying the hardness from the face of the bit backwards. This may be done by fully quenching the bit when first taken from the fire, then withdrawing it so that about only half an inch of the bit is submerged in the water, giving a slight up and down motion of about ⅛ in. to avoid the formation of a marked hardening line, and at the same time allowing the internal heat to draw the temper of the back portion of the bit. The bit will have to be held from 5 to 10 seconds in this way, and then allowed to cool right out.

(*l*) *Composition.*—It is well here to refer again to the effect of composition, as the general composition of a mining steel is somewhat dangerously near a change in the type of steel. It is advisable not to have the carbon content too high; .9 % of carbon with a low manganese content denotes the changing point in this type of steel. Steels below this carbon content have crystal boundaries of pure iron, which is soft and cushiony, and helps the material to withstand shock. Steels above .9 % carbon have a different type of boundary around the steel crystals, this boundary being composed of a very hard and brittle carbide of iron, which has to be modified by a more or less complex treatment before tools of a really satisfactory nature can be produced from this type of steel.

(*m*) *Shanks.*—

1. Soft shanks, causing burring of the face which comes in contact with the piston hammer of the machine, and hence jamming in the chuck bushing.
2. Hard shanks, causing undue brittleness in the shank itself, which should have a certain amount of resistance to shock and bending stresses. Hard shanks are also liable to damage the hammer pistons.
3. Brittle shanks—brittleness induced by over-heating, causing breakage of the shanks.
4. Shanks hardened too far back and being left in the hardened state in those portions of the shank which have to resist bending stresses.

5. Shanks forged at too low a temperature, having inherent clinks or small cracks, which are propagated by both heating and working. The writer was fortunate enough, while on the field, to collect shanks showing all these defects. Several of the shanks showed defects undoubtedly starting from initial cracks originating at the centre hole.
6. Other shank breakages may be caused by sharp roots or angles on the shanks themselves, or by tool marks forming sharp indents on the shank.

(*n*) *Fatigue.*—This may be described as an over-stressing of the steel, causing local plastic deformation of the crystals of the metal, which, when cumulative, will finally result in fracture. Under normal working conditions of the machine the writer does not consider the duty sufficient to cause the failure of the steel. In some cases this condition may be brought about by abuse or careless use of the steel in the machine, but when one examines the percentage of breaks along the length of the steel, and the numerous other causes which may lead to this particular type of break, apart from the general fatigue of material, one must be led to the conclusion that fatigue, under normal circumstances, is not a serious factor. This condition may be brought about by—

1. The use of bent drills, causing compound bending and compression stresses of an alternating nature on the steel.
2. The use of unduly worn chucks
3. Striking the steel off the square.
4. The use of unduly blunt drills.
5. Undue loss of gauge, causing jamming of the following drills.
6. Hammering the drills when jammed to release them from the holes.
7. Concentration of stresses at points where clinks, slag inclusions, segregations of impurities, sharp indented marks, or portions of the steel worked in the blue brittleness zone, and hence being internally strained, occur.

Many of these causes enumerated here are not normal to the ordinary run of work.

It is useful to note that, on questioning the underground men at the machines about the breaks along the length of the drill, the opinion seems to be pretty unanimous that these breaks mainly occur when the drills have been jammed in the holes.

Another cause of undue blunting and extra strain thrown on the machines and on the steels is the use of too short a stroke on the drill —*i.e.*, the drill is jammed up too hard against the face. This is apt to cause a grinding action on the bit faces.

Another possible cause of this fracture along the length of the bar is the unsuitable initial condition of the steel as it is received from the manufacturer.

(*o*) Undue loss of gauge. This is due to—

1. Soft bits.
2. Lack of uniform hardening.
3. Use of unduly blunted drills.
4. Decarbonization of the steel caused by long soaking at high temperature.
5. Insufficient depth of hardness.
6. The non-uniform temper of the steels themselves.

(*p*) Varying tempers for different ground. This renders all gauging data practically unreliable, since steels of the various tempers are apt to get mixed.

(*q*) Defects caused by negligent inspection of blunts arriving back at the drill sharpening shops—*i.e.*, clinks, small cracks, laps, etc., which may not be noticed on the bits. When the bits are re-heated for forging or re-dressing these defects spread, and are largely hidden again by the forging operation. They are then quenched and further propagated, with the result that, on going below, probably the wings will be fractured or the cutting faces chipped, and, again, the boring speed is impaired and extra load thrown on the steel and the machine.

(*r*) Lack of efficient inspection of the hardened drills before going underground—*i.e.*, defects which will cause irregularities in the gauge, causing jamming, etc., such as unduly soft bits, incorrect gauging, etc.

(*s*) Lack of efficient education of smiths and the lack of appreciation of the fundamentals of heating and cooling.

While Fig. 14 shows the effect of various treatment on the cutting efficiency of the drill bits, Fig. 16 gives a very clear indication of the effect of heat on the relative structure and hardness of this steel, and should be very carefully studied. It will be seen that a maximum hardening effect is obtained between temperatures of

1380° and 1440° Fahr. or 750° to 780° C.—*i.e.*, from a dullish to a medium red colour. It will be seen that hardening at this temperature

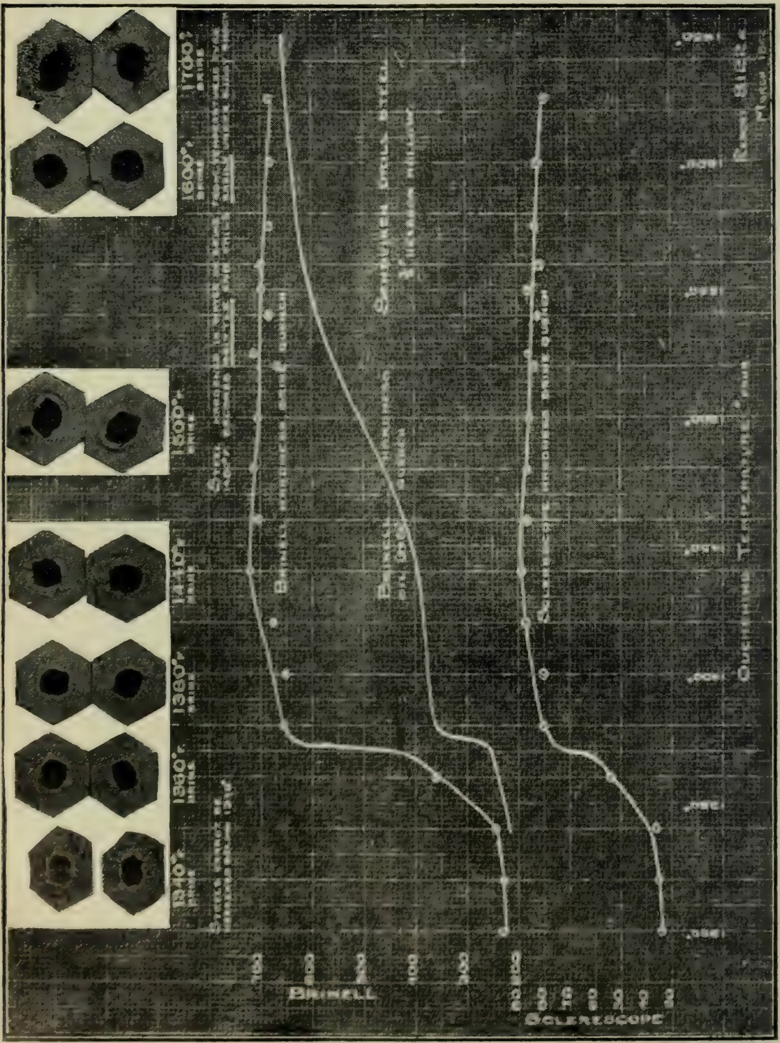


FIG. 16.

SHOWING THE PROPERTIES OF MINING STEEL WHEN QUENCHED FROM VARIOUS TEMPERATURES.

By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

gives a structure which is hard on the outer cutting edges and extremely fine, while the inner structure is of a softer and tougher nature, and supports the harder outside structure. If the temperature is raised much above 780° C. the whole structure becomes hardened throughout and the crystal size is increased, and hence made more brittle. This diagram clearly indicates that there is nothing whatsoever to be gained by raising the temperature beyond, say, 1450° Fahr. or 780° C., and after passing this temperature the steel begins to deteriorate.

(*c*) SUGGESTED PROCEDURE FOR MANIPULATION.

While the writer appreciates the fact that for many years past mining people have been able to carry on quite satisfactorily with their work, he suggests that simple modifications of many existing practices may be effective in reducing their steel casualties and giving them greater boring efficiency, at the same time reducing the depreciation of both machines and the steel. In giving these suggestions the writer has in view the economic factor, and is endeavouring to help towards getting the desired results with the least possible initial expenditure.

The first important question is the lighting of the shop in which the steel is being worked and treated. If the steel is being done by men's judgment instead of under pyrometric control, the writer is of opinion that perfectly satisfactory results may be obtained by attention to the following suggestions:—

1. If uniform results are desired from the forging and hardening operations it is essential that the light of the shop be kept as constant as possible, and on the dullish side, both in the neighbourhood of the heating furnace and the quenching baths. This may best be effected by putting bags or shutters over any windows which throw direct light either on the furnaces (where the heating is being done) or across the baths (where the quenching is being done). This is the only way of ensuring the man's judgment being uniform within reasonable limits.

2. *Fuel.*—In the writer's opinion the most suitable type of fuel to use at the present time is coke. The size of the coke used in the little forges to be recommended later needs attention. The lumps should be round, about the size of fairly large walnuts, the idea being

not to have any black lumps in that portion of the fire where the bits are being heated, as these may cause uneven heating.

3. *Inspection of Steel.*—In dealing with new steel direct from the store, this should be carefully inspected for any steel defects enumerated under Section B.

4. *Cutting to required lengths.*—This should be done by means of a power hack saw, and if the lengths required are measured off it is practically unnecessary to take the binding wire off the bundles, as the bundles of bars may be put under the saw and a boy could set the machine to work for each batch cut off, and carry on with other work while the saw was cutting through the bundle. This method of cutting off is practically essential to avoid the clinks which are liable to be formed by the method of cutting with cold set and fracturing. These defects, as previously explained, are apt to be a source of nuisance through the whole life of the drill.

5. *Inspection of blunts.*—Blunt bits coming from underground should be carefully inspected before putting into the furnace for heating up for the forging operation. From this inspection some idea can be formed of the effectiveness of previous treatment, and any cracks or defects, such as laps in the steel, etc., should be cut out before the bits are re-dressed, otherwise they are apt to be propagated as previously described, and seriously impair the cutting efficiency of the bit when returned to service.

6. *Straightening of blunt drills.*—If some of the drills are noticed to be bent on return from underground, they should be straightened before again being put into operation. This straightening should be done at a heat just slightly above the hardening heat. Sufficient time should be given for the heat to soak uniformly throughout the section of the steel. A suitable heat would be round about a medium bright red heat, and the bars, after being straightened, should be allowed to cool out in air. It is not good practice to straighten these bars at a black red or very dull red heat, as it tends to distort the steel crystals and strain the steels.

7. *Method of handling in quantities.*—This would essentially depend on the heating capacity of the furnaces. The writer strongly recommends that a pre-heating furnace of reasonable capacity be adopted for the heating of steel through the initial stages of the heat prior to the forging operation. If the heat of such a furnace is kept

at a dullish red heat quite a number of blunts may be put into this furnace to come up gradually to a medium red heat prior to heating for forging or re-dressing. The effect of this pre-heat is to allow the temper of the hardened blunt to be drawn, and so allow sufficient ductility in the metal for it to withstand the stresses due to expansion on heating. The steel soaking at this dullish heat cannot be affected harmfully, and if this pre-heating furnace were to be used as a sort of dump from which the bits to be re-heated can be taken, it means that much time can be saved in the heating up of the steel for the forging operation, and that at the same time the heating up will not be unduly rushed, and it is possible that in the long run the man may be able to get through more forging in the same amount of time. He takes the steel from the pre-heating furnace at a dullish red heat, and can bring it up to the required forging temperature from that heat fairly rapidly.

8. *Heating up for forging.*—In heating up for forging it is desirable to heat to such a temperature as will allow the forging operation to be finished at round about the hardening heat or just above it, but the heat must not be unduly high—*i.e.*, a fairly bright orange colour would be sufficient on the top side, the finishing colour being about a medium red. If the forging cannot be brought down to a finish in the one heat when starting from this orange colour, it will then be necessary to re-heat it for finishing, and if there is not much forging left to be done in the finishing heat it will be an advantage in this second heat not to heat any higher than is necessary to allow the forging to be finished at a medium red colour. Give sufficient time for the steel to become uniformly heated throughout the mass. If for any reason a forging machine is put out of action, and delay is caused, do not let the steel soak in the forging furnace, but withdraw it and place it back in the pre-heating furnace, as soaking at high temperature coarsens the grain of the steel, causing undue brittleness, and also decarbonizes the outer surface, causing undue wear of gauge and soft cutting edges.

The writer is of opinion that it may be a fair thing to put a batch of 25 to 30 drills in the pre-heating furnace at the one time, working them over in about sixes to the forging furnace.

9. *The forging operation.*—In forging, first see that all the tools are in good order, and that there are no unduly sharp edges or shoulders

on the tools. Start the forging of the bits with a few light taps to start the flow of the metal, then give it the full force the machine is capable of exerting. It is generally considered advisable to keep these forging machines up to the full 80 lb. per sq. in. of their pressure.

10. *Re-heating or normalizing.*—To reduce forging strain and to refine the grain of the metal, and so put it in its toughest condition, it is advisable after forging to re-heat the parts that have been subjected to the forging operation to a colour just slightly above the hardening colour—*i.e.*, about a medium red. Allow sufficient time for the heat to soak uniformly throughout the mass of the steel, plus a further soaking of about 10 minutes, then withdraw from the furnace and allow to cool in air. This is rather a refinement, and may be dispensed with if the limits of heat in the forging operation are carefully observed—*i.e.*, if the forging is finished at round about the hardening temperature and allowed to cool down, at any rate, to a black heat in air. On the other hand, if the forging has been finished too hot, the grain will be fairly coarse, and the material not as resistant to shock. This coarse structure may be refined by the normalizing process, which will give the maximum shock-resisting properties. Again, if the forging has been finished at too low a temperature—*i.e.*, below the hardening heat—the steel grains will be distorted, and there will be an internal strain in the metal. The normalizing operation modifies the grain and relieves the inherent stresses. The same remarks apply to the forging of the shanks. It is specially advised that the shank collars be not at any time forged below the hardening temperature, as the metal around the centre hole of the steel is apt to be distorted, and undue stress set up in the neighbourhood of the inside of the collar. The writer is of opinion that it might be a wise precaution to normalize these shanks in the neighbourhood of the collar preparatory to hardening if any considerable percentage of shank breaks has been noticed. After the normalizing operation it is not necessary to allow the steel to cool right out before heating up for hardening; but, so long as the heat has passed through a black or black-red stage, it may be heated up for the hardening operation.

11. *Heating for hardening.*—When heating for hardening special precautions should be taken. It is necessary that the sharpener should at all times be able to keep the bits that are being heated

under close observation. This is one of the reasons why the writer recommends the small coke furnace in preference to the ordinary smith's open forge fire. Another is that more drills can be dealt with at the one time in the coke fires, and with greater efficiency. It is a good scheme to have the temperature of the hardening furnace at a medium heat, so that the risk of overheating is not unduly great. The front part of the fire wants to be well banked up, so as to throw the heat evenly on the bit faces. It is a good scheme to have the fire a little dull on, say, the right-hand side, and have a quantity of drills on the front edge of the furnace heating up gradually through the initial stages of the heat, and work the drills across to the left-hand side in batches of about half a dozen, allowing the temperature to rise gradually to what is recognized as the best hardening temperature. Allow sufficient time for the heat to soak uniformly throughout the mass of the bit and evenly across its face. Do not allow the heat to run too far back behind the bit; say for a distance of from ½ in. to ¾ in. will be quite sufficient if the heat is even. With a little practice with these furnaces the writer feels confident that the hardening man will be able to judge and control his temperature fairly accurately, especially if he makes use of the fracture test, to be suggested later. While heating up for hardening, keep the drills turned occasionally in the fire in order that the heating effect may be evenly distributed across the faces of the bits.

12. *Quenching out for hardening.*—There should be a sufficient quantity of quenching media to enable the quantity of work in hand to be put through without the media becoming unduly heated. On frosty mornings it is advisable to take the chill off the water that is being used. A fairly good temperature for working is approximately 30° C., and the temperature should be brought up to round about this figure before starting to quench for hardening. This is a slightly tepid heat. The bits should be quenched straight out in water. If the smith is certain that he is using the lowest temperature to give sufficient hardness, and he finds that, after observing the precautions of gradual heating, his bits are apt to chip unduly, he may then quench by dipping straight out in water, holding there for a few seconds to bring down to a black colour, then rapidly withdraw the bit from the water and quench it in whale oil, and allow it to cool right out in the oil. The writer is of the opinion that if the manipula-

tion is carefully done the straight-out water quenching should prove effective, and not unduly dangerous. On the other hand, if the drill is heated too far back behind the bit, quench straight out in water, holding there for a couple of seconds; withdraw the drill, leaving about half an inch of the bit immersed in water, give a slight up and down motion for about an eighth of an inch to prevent a sudden hardening line between the soft and the hard portion of the bit, hold there for about ten seconds, allowing the temperature to be drawn in the portion of the bit in which toughness is required, and then drop straight back into the water and allow it to cool right out.

If in heating up for hardening the bit becomes obviously overheated, withdraw it back from the fire and allow the heat to go back to a blackish or blackish-red colour, then replace in the fire and allow it to heat up to the required hardening temperature. On the other hand, if the heat is not unduly high, the drill may be withdrawn from the fire and held over the quenching media for a few seconds until the heat has dropped to what is reckoned to be the best quenching temperature, then quench straight out.

The lowest possible hardening heat is the best heat to use, because it gives a maximum hardness with a minimum internal stress. The writer does not like the method of quenching by taking the drills straight from the fire and putting the bit on a tray which is only submerged for about half an inch in the water, because—

(*a*) A rapid change of section, due to the hardening, is likely to form around the hardening line.

(*b*) The heat behind the bit may draw the temper of the hardened portion to such an extent that it will leave only a very thin skin of hardened metal, which may soon wear or be chipped off, causing undue loss of gauge and early bluntness, as shown in Fig. 13*b*.

(*c*) By this method steam is apt to form at the centre hole, keeping the cold quenching media from the surface of the steel, and so prevent an efficient hardening in the vicinity of the hole, causing soft centres, giving results similar to 13*b*.

13. *Shanks*.—Shanks in the hexagonal steel which has a collar on should not be hardened for more than an inch to an inch and a half down. This heat may be taken a little bit higher than that used

for the hardening of the bits, but not much—say to a medium red. The shanks should then be quenched straight out in whale oil. Do not use cyanide, as the short amount of time available for the absorption of the carbon from the cyanide, and the lowish temperature used, do not allow any appreciable amount of carbon to be absorbed, hence very little result is got from the practice. On the other hand, if cyanide is allowed to be absorbed for a sufficient time at a sufficiently high temperature, with steel so high in carbon trouble is likely to arise. The writer is of opinion that perfectly satisfactory results may be obtained by quenching at the proper temperature in whale oil. With the shanks on the $1\frac{1}{4}$-in. Water Leyner drill, where there are just the two lugs, heat the shank evenly and uniformly to about one inch beyond the lugs to a medium red heat and quench out in whale oil. Keep the temperature as low as is reasonably consistent with giving sufficient hardness. In the forging of the shanks avoid all sharp corners and angles and the lapping of the steel or the forging in of scale, etc., as far as possible, as these are apt to be starting-points for defects which may eventually lead to failure.

14. *Method of checking temperature.*—Metcalf's method.—In order that the smith may understand what happens when the steel is heated and what may result from overheating, let him get a length of steel, say of $\frac{7}{8}$ in. section, and nick it with a cold set about every inch for a distance of, say, 6 or 7 inches. Heat it up as high as possible on the front end and allow the heat to taper back at the last nick to about a blackish heat. When the bar has had sufficient time for the heat to soak throughout its mass, withdraw it and quench in water, then break the steel at each successive nick. It will be noticed that the very hot portions have a very coarse, brittle grain, and the steel will be very easy to break at the parts where the heat was greatest, and the grain gets finer as the heat gets lower. Working backwards, the bar becomes more difficult to fracture. At the proper hardening heat the grain will be very fine and close; then, passing through this to the part that is not hardened at all the grain will come again of a coarsish nature, but will be greyish in colour and very tough to break. Take a similar bar, heat it in a similar manner and quench it out, then re-heat the bar gently through the first part of the heat to a colour that is reckoned to be about the correct hardening heat—*i.e.*, a medium red, on the dullish side. Allow

sufficient time for the heat to become uniform, withdraw from the fire and quench in water, then fracture the bar at each successive nick. The smith will then see that the grain will be fine all the way along. He will realize from this that it has been possible to restore by re-treatment the damage done by overheating. The finest and hardest grain in his previous test will indicate the best hardening temperature, and if in doubt at any time he may frequently check his hardening colour by nicking a bar and heating it up to what he reckons is a fair hardening heat, and quenching out and fracturing. By this method, in a very short time, these men may become excellent judges of the temperatures required for hardening this type of steel, especially if the lighting precautions, as previously advised, are carefully observed.

Some idea of Metcalf's test will be gained by studying the fractures shown in Fig. 16. A good illustration of the effect of varying temperature is also shown in Alling's modification of Metcalf's test, shown in Fig. 17, one side of the bar being cut through with a saw, and then the bar heated as described for Metcalf's test—*i.e.*, to a burning temperature on one end, tapering down to a temperature which is below the hardening temperature at the other end, and then quenched out and split. At the hot end a very coarse structure will be obesrved ; at the correct hardening temperature a very fine grain will be noticed ; and at the softer end the coarser, tough structure will be observed.

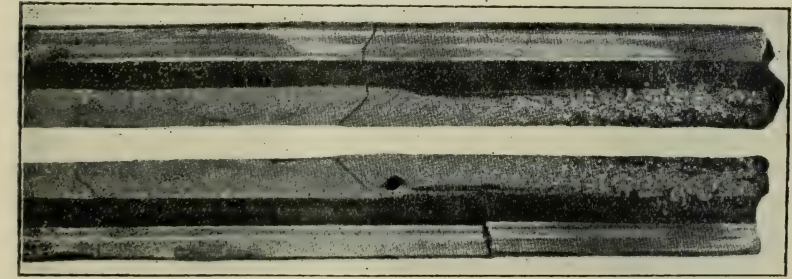


FIG. 17.
SHOWING ALLING'S MODIFICATION OF METCALFE'S TEST.
Brainerd Comp. Air Mag. By courtesy Ingersoll-Rand (Australia) Pty. Ltd

15. *Testing of bits after withdrawing from the quenching tank.*—An indication of hardness will be given by the greyish colour on the bit when withdrawn from the tank, but the writer strongly advises that a test be adopted—*i.e.*, that as each drill is withdrawn from the tank the cutting edges be tried with a small saw file for hardness. If they are unduly soft, throw the drill out for re-hardening, as a drill of this nature will cause undue loss of gauge underground, and probably cause the jamming of the following drill. A small test like this only takes a minimum of time, and is well repaid by consistent results underground.

The writer would also strongly advise the adoption of a check method for gauging, such as has been instituted on the Bendigo Amalgamated Goldfields—*i.e.*, with a small device checking the diameter of the bit with the length of the drill. On the Bendigo field a considerable amount of time was given to the question of gauging, which finally led to very satisfactory results being obtained. The testing device used there is of a very simple nature, and gives a check against the man at the forging machine. It was found that, by making use of this checking test, many of the gauging troubles were eliminated on that field. There is no question that the gauging problem is of the utmost importance.

16. *Observational training of smiths.*—Smiths should be encouraged at all times carefully to observe the blunts coming from underground for re-dressing, and to search for as much information as they can regarding the performance of the drills underground, and they should carefully correlate their observations with their steel manipulation.

17. *Marking drills.*—When marking drills at any time it is advisable to avoid marking with sharp indents or punched marks along the length of the drill, as such marks are apt to form starting-places for defects.

18. *Collection of underground data.*—The writer strongly advises the collection of underground data of an accurate and reliable nature, giving percentages of the various types of defects. It may then be easy to rectify these in the sharpening shop. A suggested field card for the collection of such data will be found on Page 127. Researches of this nature will give much valuable information about boring operations in general, and will certainly lead to a cutting-down in working costs.

(*d*) SUGGESTIONS REGARDING PLANT.

1. During various investigations of this problem on various fields the writer came to the conclusion that the lighting of the shops in the vicinity of the forging and heating furnaces was not receiving sufficient attention. As previously pointed out, in order that the man's judgment may be sufficiently accurate to carry out the manipulation as recommended in section C, it is necessary that the lighting be a first consideration. It should be kept as uniform as possible, and of a dullish nature. This may be easily arranged by the use of shutters, or, say, bags hung over the windows that throw direct light on to the forging furnace and quenching baths.

2. Ventilation of shops is of importance, and the writer would recommend that in these shops ventilation be mainly effected in the roofs of the shops, as this allows the heat to escape upwards instead of being deflected down again into the shop.

3. Floors should be kept dry in parts where hot steel is thrown about, and should be of a softish nature, so that falling steel cannot be damaged.

4. *Heating equipment.*—In no case has the heating capacity on the various fields visited appealed to the writer as being adequate for the work required. If this capacity is restricted it leads to a rush of heating, which is very undesirable. In most cases the writer has found the smith's open hearth in use for heating both for forging and hardening. Only on two fields has the writer found anything of a really adequate nature in use. It might be pointed out that with the ordinary smith's hearth the amount of steel that can be heated at one time with safety is very limited. The attention required at the fire is much greater, and it is very much more difficult to judge the temperatures, since the important parts of the drills are mainly covered by coal or coke in these fires, and the drills have to be withdrawn to judge the heats. This method of heating is very primitive, and it is unreasonable tc expect smiths to get uniform results from it. The writer is of opinion that the most suitable type of heating furnace for this kind of work is the small coke furnace shown in Fig. 18, or in operation in Fig. 19. This is a small coke furnace with a wind box under the coke bed. It is very easy to regulate and keep at a comparatively regular temperature, inexpensive in cost, and has the advantage that the important portions of the drills being heated

in it can be kept under observation the whole time. Further, many more drills may be handled with safety in this furnace in the same time than with the open hearth. The writer is of opinion that this type of furnace is eminently suitable for this particular type of work, and would suggest that two of these furnaces be provided for each sharpening machine, one being used at a low temperature as a kind of dump into which hard blunts for re-dressing, new steel for being forged up, etc., may be put in quantities to heat up gently to a dullish temperature; the other one being used to bring the steels at the dullish temperature in smaller batches up to the required forging temperatures. A third furnace is then necessary for heating

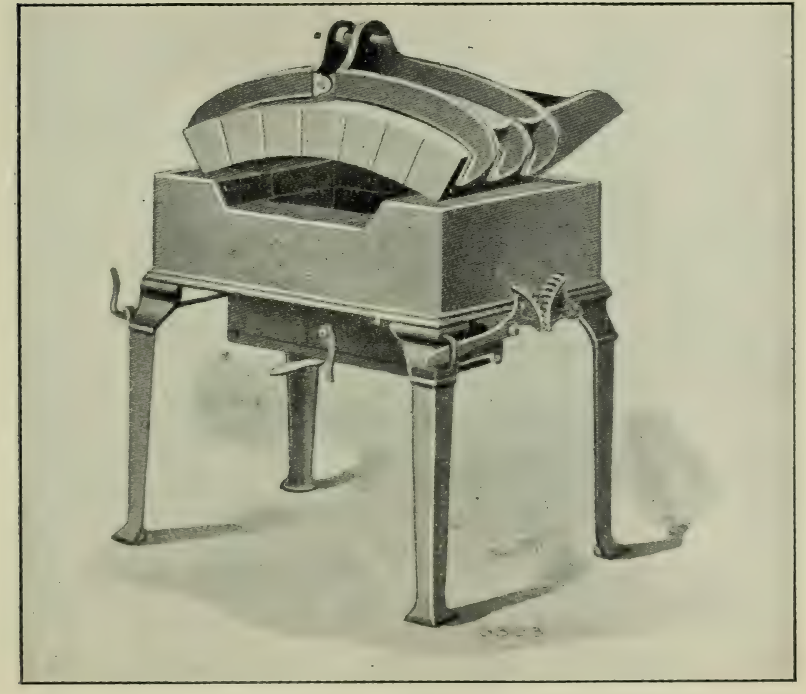


FIG. 18.
THE No. "8-B" COKE FORGE.
By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

up for hardening. If several machines are in use it is possible that the one pre-heating furnace may serve two machines, while two furnaces may be required for the heating up for forging; then one further furnace will possibly do for the hardening and tempering—that is, making three furnaces to one machine, or four furnaces to two machines. The absolute necessity for heating gently through the early parts of the heat, both for forging and hardening, cannot be too strongly emphasized; therefore it is absolutely necessary if good results are desired that there be sufficient heating capacity to heat the steel in this manner so as to get the maximum efficiency from both men and forging machines. The initial cost of these small furnaces is small; the fuel cost is also small as compared with the extra boring efficiency attained from drills uniformly and correctly treated. At the same time the writer is of opinion that many more

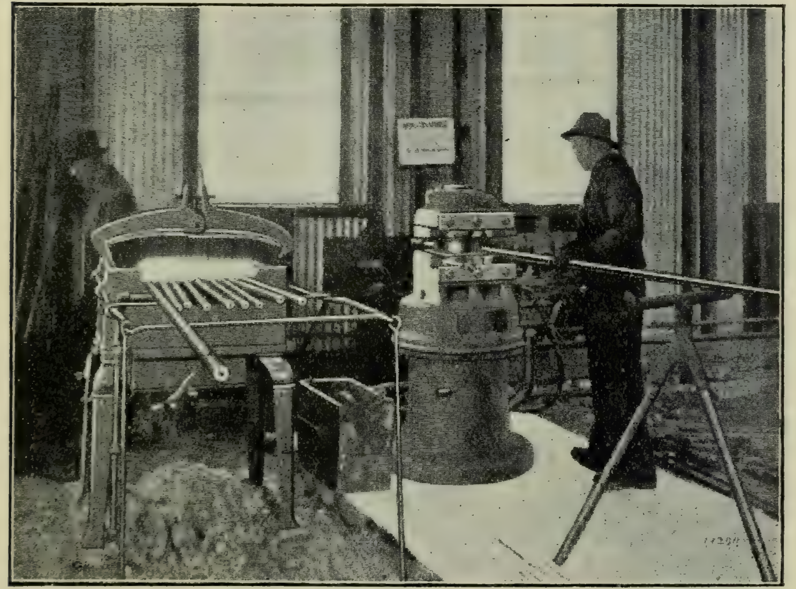


FIG. 19.
COKE FORGE OPERATION.
By courtesy Ingersoll-Rand (Australia) Pty. Ltd.

drills may be handled by this means, and, further, give far more uniform results than may be expected with the ordinary open fires. In the case of smaller shops, where the number of drills required per shift is not considerable, two furnaces may be sufficient, with one sharpening machine—*i.e.*, one used as a pre-heating furnace and the other used for the final heating up for forging, the forging being all taken through in succession, then one of the furnaces, or both, being used later for the hardening and tempering.

5. *Quenching media.*—It is essential that there should be a sufficient quantity of this to avoid it becoming overheated by the quantity of work that is required to be put through it in a given time. The writer would suggest that where oil is required, only whale oil be adopted as the standard quenching oil.

6. *Fuel.*—The writer is of opinion that coke will be found the most suitable.

7. A small coke crusher will be found a great labour-saving device, at the same time giving a much more uniform size of fuel.

8. *Forging tools.*—The writer would suggest that in many cases, such as for dollies, hole-punching needles, etc., high chrome or stainless steel be used, because it is a heat-resisting steel, and does not scale so readily at high temperatures as the bulk of other steels, and at the same time possesses a certain amount of red hardness. It is strong, hard, and tough, and, owing to its non-scaling properties, will retain far cleaner surfaces than the ordinary steels, thus giving far cleaner faces, corners, edges, etc., on the drills.

(*e*) SUMMARY.

The writer is of the opinion that considerable saving might be effected by—

1. The slight modifications of the various practices in use as suggested under Section C, "Manipulation."

2. Selection of steel from the most reliable source of supply, and standardization of the steel giving best results.

3. Careful inspection of new stocks of steel.

4. Careful inspection of blunts on return to the sharpening shop for re-dressing.

5. Cutting off defective bits so as to prevent the propagation of defects in re-heating for hardening, etc.

6. The straightening of drill bars when necessary. This to be done at a medium red heat.

7. New steel being cut to length for drills in a hack saw machine, and not by the method of cutting with cold set and fracturing. This to avoid clinks and starting-points for defects that may be propagated throughout the length of the drills in subsequent re-dressings.

8. Collection of reliable data of a quantitative nature will be found to be of great assistance in the solving of treatment and boring problems.

9. It is necessary to have sufficient heating capacity for dealing with the required amount of drills without prejudice to the correct rates of heating and recommended methods of treatment.

10. Always heat gradually through the initial stages of heating.

11 It is necessary that heating be of a uniform nature and evenly distributed throughout the mass of the steel.

12. Attention to lighting conditions in the forging and hardening shop is of importance ; this to be as constant as possible, to ensure uniform judgment.

13. Ventilation important, and best effected from the roof of the shop.

14. Good condition of forging tools essential, and it is necessary to avoid sharp corners at points where heavy stresses are liable to be encountered.

15. Forging heats to be within reasonable limits, and generally such that the forging will be finished at a temperature just above the hardening temperature—*i.e.*, about a medium red.

16. Over-heating (which is responsible for the majority of steel troubles) should be avoided. In forging a steel of this nature it is advisable not to exceed a bright orange colour.

17. Forging at a temperature below the hardening temperature to be avoided—*i.e.*, below a medium red.

18. The lowest temperature at which steel will harden satisfactorily is the best temperature to use.

19. It is best to use one temper only for all drills, and that temper should be the one which will stand up best to the most severe conditions.

20. The judgment of the man for hardening temperatures should be checked occasionally by the fracturing of test pieces (Metcalf's

method). The heat that gives the finest grain is the right temperature for hardening.

21. Testing the drill bits with a file as they are removed from the quenching media is advisable.

22. Eliminate all soft drills or those on which defects can be detected in order to avoid—(*a*) expense of transport down the shaft, (*b*) loss of boring efficiency, and (*c*) the possible compensation for accidents.

23. Discourage the use of unduly blunt steel underground, as bluntness prevents rock penetration, throws extra duty on the steel and the mechanism of the machine, and leads to an excessive loss of gauge.

24. Avoid the use of unduly worn chuck bushings in the machine.

25. Avoid the use of unduly worn hammers.

26. Avoid the use of bent drills.

27. Avoid the use of shanks which are not properly squared off.

28. Standardize the steel treatment on methods based on fundamentals of heat treatment, and on modifications found to be necessary and practicable in actual practice. When this is accomplished standardize the gauging of the bits.

29. AT ALL TIMES avoid forging and hardening in the one heat; the method is absolutely unreliable, and throws additional internal stresses into the steel, and will be found responsible for a large loss of boring efficiency underground.

30. The use of cyanide in hardening shanks at the present time is inadvisable.

31. Encourage the sharpener to increase his powers of observation and to correlate the data so obtained with the fundamentals of steel treatment.

32. Shanks should be hardened so that they can be readily cut with a file and yet stand up. The shanks must be softer than the piston hammers, or damage to both will result. This is best effected by hardening the shanks in whale oil from a heat slightly above that required for the hardening of the bits, which should be hardened in water.

33. It is important that there be a sufficient quantity of any quenching medium to avoid it being overheated, and hence its hardening efficiency reduced

34. The chill should be taken off the water when using for quenching, the best temperature being between about 80° and 100° F.—*i.e.*, 27° to 38° C. This recommendation coincides with what has been found the best practice at the Copper Range mines in America.

Attention to the above points will certainly reduce the casualties and at the same time substantially increase the boring efficiency of both the drills and the machines, and reduce the depreciation of machines and steel, and will in a very short time much more than compensate for the time, trouble, and expense incurred in investigating the drill-steel and boring problems.

The writer is of opinion that the use of pyrometers, electric furnaces, etc., is rather too much of a refinement for the average mining field, and, as practically only one type of steel is being used for drilling purposes, it is possible, by careful observation based on fundamentals, to get quite good results.

(*f*) SUGGESTED RESEARCH AND TABLE FOR RECORDING DATA.

The writer would suggest that there be some scheme of collecting underground data recording the percentages of failures and the cutting efficiency of the bits, also giving reliable figures on which to base the gauging allowance after the treatment has been standardized, and the bits are found to be fairly uniform in use. A suggested tabulated form for the collection of this data will be found on Page 127. Figures of this nature would be of very great value in showing what sort of efficiency may be reasonably expected from the steel, and also how the rate of boring is affected. The main points of interest are as under:—

1. The brand of steel.
2. Its size and section.
3. The type of machine in which it is used.
4. The carbon and manganese contents of the steel.
5. The depth drilled in inches for each drill.
6. The time taken in getting to such a depth.
7. The initial diameter of the bit when put into the machine.
8. The final diameter when taken from the machine.
9. The loss of gauge.
10. The nature of the ground.
11. The location of the job.

SUGGESTED TABLE FOR COLLECTION OF DATA.

Brand of steel
Type and number of machine
Nature of ground
Dates

Size and section
Location of tests
Analysis { Carbon content =
Analysis { Manganese content =

Number of hole.	Number of drill.	Length of drill.	Initial depth of hole in inches.	Depth drilled in inches.	Time in mins. and secs.	Average time in inches per min.	Initial diam. of bit.	Final diam. of bit.	Loss of gauge.	Condition of bit at finish.					Condition of shank.				Other breakages.			Remarks.
										Chipped.	Blunt end turned over on wings (soft).	Soft centres.	Even and regular.	Uneven wear from centre to outside, with rounded off corners.	Soft and burred	Chipped.	Broken.	Good.	At collar.	Behind bit.	Along length of drill.	
1	1	2 6	Nil	12	2	6	$1\frac{7}{8}$	$1\frac{13}{16}$	$\frac{1}{16}$				1					1				Satisfactory,
1	2	3 6	12	12	2	6	$1\frac{13}{16}$	$1\frac{3}{4}$	$\frac{1}{16}$				1		1							Shank too soft.
1	3	5 6	24	24	3	8	$1\frac{3}{4}$	$1\frac{11}{16}$	$\frac{1}{16}$				1					1				Good.
1	4	7 0	48	6	1	6	$1\frac{11}{16}$	$1\frac{5}{8}$	$\frac{1}{16}$			1						1			1	Jammed and hammered to release from hole.
1	5	7 6	54	15	3	5	$1\frac{5}{8}$	$1\frac{17}{32}$	$\frac{3}{32}$	1								1				Fracture coarsish, indicating high hardening temp.
2	6	2 6	Nil	12	2	6	$1\frac{7}{8}$	$1\frac{13}{16}$	$\frac{1}{16}$				1					1			1	Sharp indented mark at point of fracture.
2	7	3 6	12	12	3	4	$1\frac{13}{16}$	$1\frac{11}{16}$	$\frac{1}{8}$		1							1				
2	7			93	16	6				1	1	1	4		1			6			2	
										14	14	14	58		14			86			29	Percentage.
Number of holes bored.	Number of drills used.			Total length of boring.	Time in mins. and secs.	Average time in inches per min.			Average loss of gauge per foot run.	Number of chipped bits.	Number of soft bits.	Number of soft centres.	Number of satisfactory bits.	Number of unevenly worn bits.	Number of soft shanks.	Number of chipped shanks.	Number of broken shanks.	Number of satisfactory shanks.	Breaks at collar.	Breaks behind bit.	Breaks along length of drill.	Summary.

12. The condition of the bit after use—*i.e.*, whether it is chipped, blunt, or turned over on the wings, soft-centred, evenly and regularly worn, unevenly worn from centre to outside, with rounding off at the outside of the face and undue loss of gauge.

13. Condition of shank—*i.e.*, whether soft, chipped, or broken.

14. Other breakages, whether at the collar behind the bit or along the length of the steel that has not been subjected to heat in the hardening shops.

15. A "remarks" column, giving any information relating to any particular breakage—*i.e.*, if the drill were jammed in the hole or if it were hammered to release the bit; whether the chuck on the machine were loose; whether the drill was bent; the nature of the particular break; whether any flaws were obvious; whether any sharp indented marks were noticed at the point of fracture; whether the drilling speed seemed to be unduly reduced owing to blunt bits, and a summary at the bottom of these columns from which the percentages may be got. It is realized that it is necessary to eliminate all writing as far as possible in the making of notes, and it is suggested that some printed form, such as the one submitted on Page 127, be supplied, and that figures and ticks under particular headings may serve to indicate the nature of various defects, etc., with just a very brief note in the "remarks" column.

The writer suggests that some such collection of information will be found of immense value, and will at the same time be an excellent check on the efficiency of the sharpening shop.

PROCEEDINGS AUS. I.M.M. (INC.)] [N.S., No. 52, 1923.

AN OUTLINE OF THE ECONOMIC GEOLOGY OF TASMANIA.

BY LOFTUS HILLS, M.B.E., D.Sc.

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2. GEOLOGICAL DISTRIBUTION.
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I.—INTRODUCTION.

THIS paper is not intended to be more than a mere outline of the economic geology of Tasmania. It is desired to present in a condensed form the present state of our knowledge in regard to the distribution of the ore, coal, and oil-shale deposits of Tasmania in time and space.

Tasmania is exceptionally well endowed with mineral wealth, particularly in the north-western and western districts. In the enthusiasm resulting from the first discoveries of mineral deposits at a great number of localities in those districts, it became a generally accepted belief that the whole of that portion of Tasmania constituted a continuous metalliferous province. As geological research in these areas has progressed, however, it has become increasingly clear that ore-deposition has been confined on an economic scale to definite well-defined areas or belts which have a genetic relationship to certain tectonic and petrologic phases of the general geology. These areas or belts have recently been defined by the writer, and in this paper they will be concisely indicated and their geographic and geologic relationships will be dealt with from the economic standpoint.

Opportunity will also be taken to present very concisely the salient points in regard to the coal and oil-shale deposits of Tasmania which have become apparent as the result of investigations carried out during the last three years under the writer's direction.

The accompanying map of Tasmania (Fig.1) shows the geographical distribution of the mineral, coal, and oil-shale resources.

As bearing on the industrial future of Tasmania, the relationship between the distribution of the ore, coal, and oil-shale resources of the State and its valuable hydro-electric power resources will be dealt with in a general way, but not in detail.

II.—ORE DEPOSITS.

1.—Geographical Distribution.

In order to focus attention on the ore deposits, the belts or areas containing deposits of economic value will be indicated. These are the metalliferous provinces, and it must be remembered at this stage that in considering them the question of metallogenesis is not taken into account. The view-point at this stage is purely geographic, combined with economic importance. The genetic aspect will be considered later.

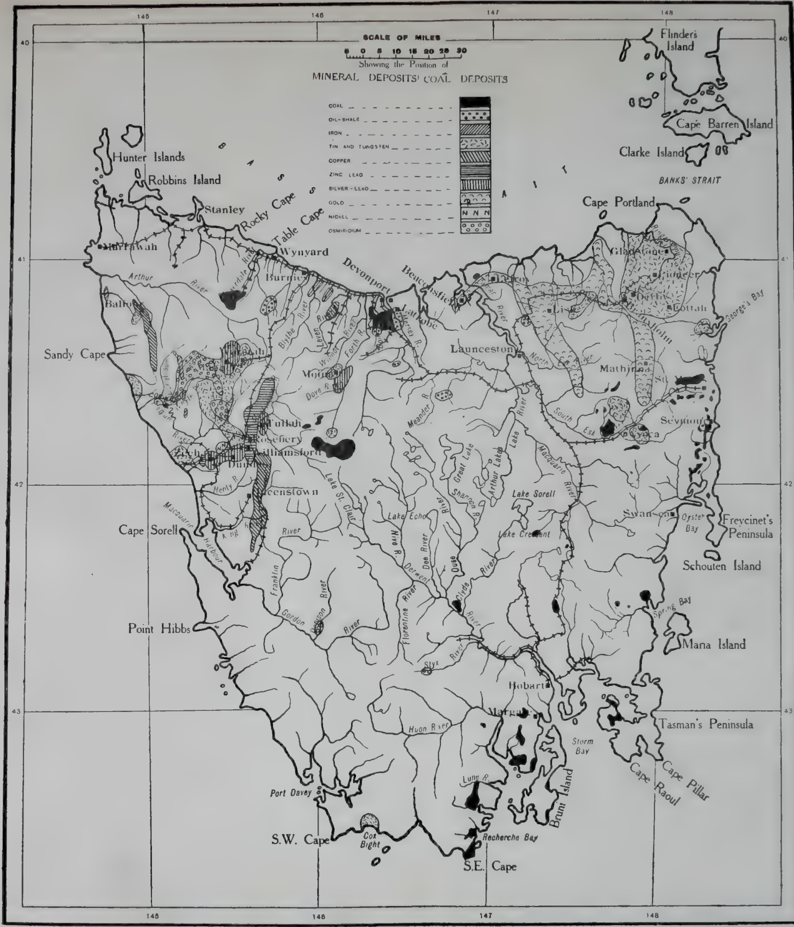
The provinces are grouped under the various metals as follow :—

TIN—TUNGSTEN—BISMUTH.

North-Eastern Province.—This is the largest of the tin provinces, and contains such well-known deposits as the Briseis, Pioneer, Gladstone, and Anchor. The characteristic type of deposit is alluvial, either as deep leads or terraces. Primary deposits are, with the exception of the Anchor, quite subordinate. This fact will be referred to at a later stage in this paper when discussing the significance of the genesis. Tin is predominant, tungsten rare, and bismuth absent.

Ben Lomond Province.—Although small in area, this is an important province, including within it the Storey's Creek, Gipp's Creek, Mount Rex, and Republic mines. Primary deposits pre-

TASMANIA.


SCALE OF MILES
5 0 5 10 15 20 25 30
Showing the Position of
MINERAL DEPOSITS; COAL DEPOSITS
COAL
OIL-SHALE
IRON
TIN AND TUNGSTEN
COPPER
ZINC LEAD
SILVER-LEAD
GOLD
NICKEL
OSMIRIDIUM
145
146
147
148
40
41
42
43
Flinders Island
Cape Barren Island
Clarke Island
BANKS' STRAIT
B A S S S T R A I T
Hunter Islands
Robbins Island
Stanley
Rocky Cape
Table Cape
Wynyard
Burnie
Devonport
Beaconsfield
Latrobe
Launceston
Cape Portland
Gladstone
Pioneer
Derby
Lottah
Mathinna
Scottsdale
George's Bay
Sandy Cape
Balfour
Arthur River
Blythe River
Leven River
Wilmot River
Forth R.
Mersey R.
Dove R.
Tamar River
North Esk River
South Esk
Meander R.
Tullah
Rosebery
Williamsford
Zeehan
Dundas
Queenstown
Henty R.
Macquarie Harbour
King R.
Cape Sorell
Lake St. Clair
Great Lake
Arthur Lakes
Lake Sorell
Lake Crescent
Lake Echo
Shannon R.
Dee River
Nive R.
Ouse River
Clyde River
Derwent River
Florentine River
Franklin River
Gordon River
Denison River
Styx River
Macquarie River
Lake River
Fingal
Avoca
Seymour
Swansea
Oyster Bay
Freycinet's Peninsula
Schouten Island
Spring Bay
Maria Island
Point Hibbs
Hobart
Margate
Storm Bay
Tasman's Peninsula
Cape Pillar
Cape Raoul
Bruni Island
Huon River
Lune R.
Recherche Bay
S.E. Cape
Port Davey
Cox Bight
S.W. Cape

dominate, and alluvial is small in amount. Tin and tungsten are about equally distributed; bismuth is absent.

St. Paul's Dome Province.—This resembles the former province, being a near neighbour. It contains the Royal George mine. Tin is predominant; tungsten and bismuth rare.

St. Helens Province.—Alluvial deposits characterize this province, which includes the Argonaut mine. Tin only is present.

Scamander Province.—This is a small metalliferous province in which primary deposits form the sole type. The Pyramid mine is included in this province. Tungsten predominates over tin, and bismuth is absent.

Springfield Province.—Whilst containing no deposit of importance, both primary and secondary tin deposits occur in this area to a sufficient degree to give it the dignity of a metalliferous province.

Dundas-Stanley River-Cleveland-Bischoff Province.—Although constituting one continuous metalliferous province, this area includes within it ore deposits possessing a wide diversity of form and mineralogic composition. Thus, there are the metasomatic replacement deposits of Renison Bell and Stanley River, characterized by massive pyrrhotite with disseminated cassiterite,* the typical contact-metamorphic deposits carrying cassiterite at Mount Lindsay,† the typical fissure-fillings of the Bischoff Extended and the Queen and other lodes at Bischoff, and finally the huge residual or detrital concentrations of Bischoff itself. With the exception of the Bischoff Brown and White Faces, etc., the more important deposits of this province are primary. Tin predominates, and tungsten and bismuth are rare.

Heemskirk Province.—With the exception of the Tamar River sub-basaltic alluvial at North Heemskirk, the ore deposits of this province are primary. Such mines as the Federation and Mayne's are included in this area. Almost wholly tin; very little tungsten, and occasional bismuth

Balfour-Pieman Province.—At present shown as disconnected, this is really one province. The deposits are wholly primary lodes. Tin and tungsten equally distributed; bismuth absent.

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* Ward, L. K., Geol. Surv. Tas., Bull. 6; Waterhouse, L. L., Geol. Surv. Tas., Bull. 15.

† Waterhouse, L. L., Geol. Surv. Tas., Bull. 15.

Middlesex Province.—Although small in area, this province is important, containing as it does the only valuable deposit of bismuth so far discovered in this State. The Shepherd and Murphy mine produces tin, tungsten (wolfram), and bismuth; the Lady Barron mine, tungsten and bismuth; and the neighbouring mines contain one or more of these metals. A little alluvial occurs, but the deposits are predominantly primary.

Pelion Province.—Primary deposits characterize this small province, which has not been much developed. Tungsten predominates.

Cox's Bight Province.—Situated on the extreme southern coast, this province is far removed from the main areas of ore-deposition. Alluvial deposits are the only type so far found, although fragments of lodes have been discovered. Tin only is present.

COPPER.

Copper has been found at a large number of points in Tasmania, but, with the exception of four areas, such deposits are so small and the copper content so low that they are unlikely ever to be of any economic value. The four areas, however, constitute four copper provinces, as follow:—

West Coast Range Province.—Extending along the whole length of this mountain range, this province contains the most important copper deposits of the State, including as it does the Mount Lyell and North Lyell deposits. Many varieties of deposits occur within this province, but they are almost wholly metasomatic replacements. The cupriferous pyrite deposit of Mount Lyell and the bornite-chalcocite-chalcopyrite deposits of North Lyell are well known. Besides these there are the cupriferous schists, which are disseminations of pyrite and chalcopyrite in schist. These are of rather extensive occurrence. In addition, there are the extensive hæmatite and magnetite deposits, carrying pyrite and some chalcopyrite, which characterize the southern and northern ends of the province in the Jukes-Darwin and Red Hills areas respectively.* All of these deposits are of primary origin, secondary enrichment having taken place only on a limited scale.

Balfour Province.—The Murray Reward mine and the associated lodes are included in this province. Secondary enrichment was responsible for the comparatively high copper values of these fissure lodes, which were not maintained when water-level was reached.

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* Loftus Hills, Geol. Surv. Tas., Bulls. 16 and 19.

Leven River Province.—Up to the present no deposit of importance has been located in this area, but there exist a sufficient number of deposits carrying appreciable copper to warrant its inclusion in the copper provinces of Tasmania.

Scamander Province.—Situated on the east coast, this small province is of little economic importance. It is included here because one mine—the Orieco (formerly Eastern Proprietary)—has been repeatedly worked because of a localized enrichment due to secondary processes. As will be pointed out later, the consideration of genesis shows that copper is not the dominant metal in the metallogenic province which controls this area.

SILVER—LEAD.

Zeehan-Dundas Province.—This is the most important as well as the most extensive silver-lead province of Tasmania. It extends from the Comstock on the west through Zeehan to Dundas, and extends to the Ring Valley on the east. It includes all the well-known galena and fahl-ore mines of this area. The ore deposits are wholly primary, and are predominantly fissure fillings.

Mount Farrel Province.—Another important silver-lead province is that in the Mount Farrel district, which includes the North Mount Farrel mine. The lodes are similar to those at Zeehan.

Magnet Province.—Grouped around the Magnet mine occur a series of deposits characterized by galena in a dolomite gangue. These constitute a definite lead province, and include the Mount Jasper and Whyte River mines.

Mount Claude Province.—The Round Hill mine is the chief of a group of lodes extending southwards towards Lorinna which constitute another silver-lead province. The Devon mine is included in this province. The lodes are similar to the Zeehan occurrences, but at the Round Hill they occupy anticlinal saddles.

ZINC—LEAD—SILVER.

Read-Rosebery Province.—The Read-Rosebery zinc-lead sulphide belt contains some of the most important ore deposits of the State. It actually extends northwards of Rosebery through the Chester and Pinnacles district to Mount Block, in the north. The ore-bodies are metasomatic replacements of calcareous schist, and are all of the same general type, although the Chester deposit is almost wholly pyrite.*

Comstock Province.—Immediately to the east of Heemskirk, in the Comstock district, a group of ore deposits occur, in which zinc and lead occur accompanied by silver. Although of small extent, this area constitutes a zinc-lead-silver province of economic importance. They occur both as replacements and fissure fillings.

IRON.

Long Plains Province.—In a narrow belt of country crossing the Savage River, on the Long Plain, and extending southwards towards the Pieman, there occur large magnetite deposits. These constitute a definite iron province which is perhaps the most important in Tasmania. In places sulphides are associated with the magnetite, and in the early days these occurrences caused mining for copper to be undertaken. Hence the names Rio Tinto, Tharsis, etc., applied to mines which are essentially iron mines. The ore-bodies are large lenses in amphibolite schist.

Comstock Province.—Close to the border of the Heemskirk granite there occur in the Comstock region numerous bodies of high-grade magnetite. Although small in extent, this is an important iron province.

Blythe-Penguin Province.—This well-known province contains hæmatite deposits in the form of irregular ore-bodies. The ore is free from sulphides, but is somewhat silicious.

Beaconsfield Province.—Associated with the serpentine in this district, iron ores, which are mostly residual concentrations, occur. In some of these chromite from the serpentine has become included, and the result is a chromiferous limonite or hæmatite carrying 2 to 3 % Cr.

Balfour Province.—On the Nelson River, in the northern part of the Balfour field, a deposit occurs carrying rich hæmatite. It is doubtful whether this area carries sufficient iron to justify its inclusion among the iron provinces, as it seems quite possible that the iron lode mentioned by L. K. Ward * is merely a local and fortuitous increase in the iron content of an essentially quartz-ankerite lode series.

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* Loftus Hills, Geol. Surv. Tas., Bulls. 19, 23, and 31; Read, A. M., Geol. Surv. Tas., Bull. 28.

West Coast Range Province.—In characteristic and extensive development there occur along this range a series of magnetite and hæmatite deposits, generally, although not invariably, accompanied by sulphides.† Iron predominates in these deposits, and there undoubtedly exists here an iron province practically coincident with the copper province indicated above. The two provinces seem to be separate in places, although they are contiguous ; but definite overlapping is observable.

GOLD.

North-Eastern Province.—Extending from Beaconsfield through Lefroy, Lisle, Golconda, Warrentinna, Mount Victoria, and Mathinna to Mangana, with a southern arm running from Lisle to Camberd Plains and the North Esk mine, this province is the only essentially gold province of economic importance in Tasmania. It includes within it such well-known mines as the Tasmania, Volunteer, Pinafore, Ringarooma, Golden Gate, and Associated mines. The character of the gold-quartz fissure fillings throughout this province is markedly constant.

Corinna Province.—Somewhat extensive alluvial gold deposits have been worked in this area, including the well-known Rocky River workings. The gold is wholly secondary, having been released by the weathering and decomposition of sulphide ore-bodies carrying small quantities of gold. In places this gold in solution has been deposited in the joint planes of sandstones, etc. In many other places on the West Coast similar alluvial gold occurs, but in every case the origin has been by secondary processes, and the amount of gold is insufficient to warrant the delineation of gold provinces.

NICKEL.

Dundas Province.—This is the only nickel province in Tasmania which is likely to be of any economic value. Nickel is also known to occur at Trial Harbour, south of the Heemskirk tin province, and at the Hazlewood, but in such small quantities that even their ultimate value is doubtful. At Dundas there occur well-defined ore-bodies carrying nickeliferous pyrrhotite, pentlandite, and chalcopyrite, and the assay values are approximately 5 % Cu and 10 % Ni. The Dundas-Cuni and other mines occur in this province.

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* Ward, L. K., Geol. Surv. Tas., Bull. 10.

† Loftus Hills, Geol. Surv. Tas., Bull. 16.

OSMIRIDIUM.

Long Plains Province.—Over an appreciable area in the vicinity of the Long Plains osmiridium is found associated with the ultrabasic rocks,* and the alluvial deposits of the rivers and creeks derived from the erosion of these rocks have been worked for some years for this mineral. The Nine-Mile Creek, Bald Hill, Savage River, Mount Stewart, are well-known localities in this connection. The mineral has been found disseminated in the serpentine, but almost all the output has been obtained from the alluvial.

Wilson River Province.—This province is separated from the former by part of a tin province, but possesses exactly the same characteristics.

Weld, Denison and Styx Provinces.—These little-known areas undoubtedly contain osmiridium derived from the serpentine outcropping in those areas, and are mentioned here as being of possible future value.

2.—Geological Distribution.

(1) THE ROLE OF THE RESPECTIVE GEOLOGIC UNITS AS REPOSITORIES OF THE ORE DEPOSITS.

Pre-Cambrian.—With the exception of one or two lodes belonging to the Pelion tin province, there has not yet been discovered ore deposits of even potentially economic value associated with the quartz-mica schists and quartzites of this rock system. It seems to be wholly devoid of valuable ore deposits. What may be ultimately discovered in the south-western portion of the State—which is largely unexplored, but which apparently consists largely of Pre-Cambrian rocks—only the future will decide, but, judging from the non-occurrence of important mineral deposits in rocks of this age in other parts of Tasmania, the prospects do not seem very bright.

At one time the magnetite deposits of the Long Plains iron province were thought to be in rocks of this age, but the writer has recently established the fact that the hornblende schists in which they occur belong to the Porphyroid Igneous Complex.*

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* Reid, A. M.

Cambro-Ordovician Sedimentaries.—The Zeehan slates and sandstones, the Bischoff slates, the Dundas slates and breccias, the Read-Rosebery schists, the Balfour slates and sandstones, and the Mathinna slates and sandstones, all belong to this system. They are the repositories of some important ore deposits.

The whole of the lodes of the North-Eastern gold province occur in the Mathinna series, as well as those tin and tungsten lodes of the Ben Lomond, St. Paul's Dome, and Scamander tin provinces, which are not confined to the granite.

The Read-Rosebery zinc-lead-silver province is confined to the Read-Rosebery schists.

Both the tin and copper provinces of Balfour occur in the Balfour slates and sandstones, and the same applies to the Balfour iron province.

The Mount Farrel lead province occurs wholly in the Zeehan slates, whereas the Zeehan-Dundas lead province embraces lodes which have been formed in this series as well as the Dundas slates and breccias, and, as will be indicated below, the spilitic and related members of the Porphyroid Igneous Complex.

The Comstock zinc-lead-silver province is wholly, and the Comstock iron province partially, in the Zeehan slates and sandstones.

The Dundas nickel province is at the contact of the Dundas slates with basic Epi-Silurian dykes.

Those lodes of the Dundas-Bischoff tin province which are not in the Epi-Silurian igneous rocks occur either in the Bischoff slates in the north or the Dundas slates and breccias in the south. Some of the Bischoff deposits, the Mount Cleveland, Mount Lindsay, and Renison Bell and neighbouring lodes are in these sedimentary series. In general it may be stated that in this large and important tin province the greater number of the ore deposits occur in the sedimentary rocks.

Porphyroid Igneous Complex Cambro-Ordovician and Epi-Cambro-Ordovician.—This most interesting series of highly metamorphosed igneous, plutonic, intrusive, effusive, and fragmental rocks contain some of our most important ore deposits.

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* Loftus Hills, "The Metallogenic Epochs of Tasmania," accepted as thesis for D. Sc. degree, to be published shortly.

Thus, the Mount Lyell pyrite ore-body is a replacement of schist, which is a highly-altered volcanic ash or tuff, whilst the North Lyell ore-bodies replace a similar schist in addition to the West Coast Range conglomerate.

The remaining deposits of the West Coast Range copper province occur either in these old volcanic tuffs, now converted into various types of chloritic or sericitic schists, or in effusive members of the Porphyroid Igneous Complex, such as keratophyre, granophyre, etc. The greater number, however, are confined to the schists.

The West Coast Range iron province follows the plutonic and effusive members of the complex, the Darwin granite being the repository of both magnetite and hæmatite deposits. The greater number of these deposits, however, occur in the keratophyre, particularly at the Red Hills, Tyndal, and Lake Dora areas.

A large number of the Zeehan lodes are contained in the spilitic and similar tuffs and lava flows of the complex.

The Leven copper province also follows a development of mostly basic effusive members of the complex.

The Long Plains iron province is confined to a narrow belt of basic rock belonging to this Igneous Complex. This rock was originally probably a gabbro intruded into the Pre-Cambrian schists, but has since, in common with the other members of the complex, suffered intense structural rearrangement and mineralogic reconstitution, and is now a hornblende schist.

Silurian.—The West Coast Range conglomerate series forms the base of this system in Tasmania, and is succeeded by the Gordon River limestone series.

The West Coast Range conglomerate is generally devoid of ore deposits, but in three localities it is the repository of valuable ore. At the North Lyell mine the bornite ore-bodies occur at the faulted junction of the porphyroid schists and this conglomerate series. The ore has replaced both the schist and conglomerate. In the Blythe-Penguin iron province the hæmatite has replaced sandstones belonging to this series, and the same action may be observed on the West Coast Range, especially in the vicinity of the Mount Lyell and North Lyell mines. At the Round Hill mine, in the Mount Claude silver-lead province, the saddle lodes occur in folded conglomerate and sandstone beds of this series.

The Gordon River limestone series carries the tin-tungsten-bismuth lodes of the Shepherd and Murphy and Lady Barron mines, in the Middlesex tin province. The limestone beds also carry some of the lodes in the eastern part of the Zeehan field.

Epi-Silurian Basic Plutonics and Intrusives.—As originally established by L. K. Ward,* the basic phase of the Epi-Silurian petrogenic period preceded by a short time interval the acid phase.

In the ultra-basic rocks, originally peridotites, the osmiridium crystallized as an original component, and is found as such in certain localities in the serpentine of the Long Plains osmiridium province. The Beaconsfield iron province is also in serpentine.

Almost all the lodes of the Magnet silver-lead province are confined to basic intrusives, such as websterite, websterite porphyry, etc.

At the Bischoff and Renison Bell mines basic dykes have been the repositories of portions of the tin lodes.

Epi-Silurian Acid Plutonics.—In the Heemskirk, North-Eastern, and Springfield tin provinces practically all of the lodes occur within the granite. This applies in a less degree to the Ben Lomond and St. Paul's Dome provinces, and in a still smaller degree to the Dundas Lindsay-Cleveland-Bischoff provinces.

Permo-Carboniferous and Trias-Jura.—With the exception of some unimportant gold deposition in Permo-Carboniferous mudstones at their contact and the alkaline intrusives at Port Cygnet, neither of these two systems carry any ore deposits of any kind.

Cretaceous Diabase.—No ore deposits of any kind are associated with this rock, with the exception of a local development of pyrite carrying a little gold at Lawrenny.

Tertiary Sediments.—The most valuable tin deposits of the North-Eastern tin province consist of gravels, etc., of Tertiary age, which ultimately were covered by Tertiary basalt. These deep leads contain tin derived from the erosion of the primary tin deposits in the granite, and are thus secondary.

Basalt.—No ore deposits are associated with this rock.

Pleistocene and Recent.—Numerous alluvial tin deposits in the North-Eastern tin province are of this age, some at least being formed by melting snow-fields or glaciers and deposited as "outwash aprons."

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* Ward, L. K., Proc. Aus. Ass. Adv. Sc., 1911.

The alluvial osmiridium is found in recent gravels and terraces, having been derived from the erosion of the parent serpentine.

(2) THE CHRONOLOGIC GROUPING.

Having very concisely indicated above the geologic period during which the ore deposits of secondary origin were formed, it now remains to present a summary of the time relationships of the primary ore deposits. In other words, it is necessary to give in outline the metallogenic epochs of Tasmania. The writer has recently dealt with this problem in a comprehensive manner in a thesis which will be published shortly. It is intended in this paper, therefore, to merely indicate the salient points of the conclusions as to metallogenesis in Tasmania.

There have been two main metallogenic epochs in Tasmania. Both of these coincided with intense orogenic movement succeeding prolonged sedimentation, and exactly synchronized with and were genetically related to the batholithic end-point of these paroxysms. Consequently there exists a close relationship between the metalliferous provinces on the one hand and the respective tectonics and batholithic invasions of the two orogenic periods on the other. These relationships are fully discussed in the paper above referred to, and nothing more than merely their existence can be attempted here.

The two metallogenic epochs will now be indicated, and the metalliferous provinces belonging to each will be enunciated.

Epi-Cambro-Ordovician Metallogenic Epoch.

Basic Phase.—This initiates the history of ore-deposition in Tasmania as far as any existing ore deposits allow us to judge. The intrusion of basicplutonics, probably gabbro, of the Porphyroid Igneous Complex into the Pre-Cambrian schists of the Long Plains brought with it abundant iron. This segregated as magnetite, which in the form of irregular lenses, survived the subsequent conversion of the basic rock into hornblende schists. A little copper and nickel accompanied the iron.

Acid Phase.—Following on the basic invasion came the irruption of the acid pole, which was accompanied by predominant iron, some copper, and a little silver and gold.

There were thus deposited the magnetite and hæmatite deposits of the West Coast Range iron province which occur in the members of the Porphyroid Igneous Complex, in close association with the granite and syenite of that igneous series. It is possible that some of the copper deposits, particularly the lowest grade ore-bodies of the overlapping copper province, belong to this epoch, but the evidence is not complete enough as yet to warrant a definite conclusion.

The Epi-Cambro-Ordovician metallogenic epoch was characterized by an excess of iron, together with some copper and nickel and a little silver and gold. It was *par excellence* the iron epoch of Tasmania.

Epi-Silurian Metallogenic Epoch.

Orogenic Phase.—During the orogenic movement vadose water, circulating under exceptional conditions of temperature and pressure, dissolved iron from the pre-existing iron deposits of the preceding metallogenic epoch. This iron subsequently deposited in the form of hæmatite either as veins in or more extensively as replacements of the conglomerates and sandstones of the West Coast Range conglomerate series. Thus were formed the hæmatite deposits of the Blythe-Penguin iron province and those of the West Coast Range iron province which occur in the conglomerate series.

Basic Phase.—The ultra-basic rocks in the Long Plains and Wilson River districts carried sufficient osmium and iridium to produce concentration of osmiridium on crystallization adequate to develop the two osmiridium provinces. The same applies to the Weld-Denison-Styx provinces. These rocks elsewhere in Tasmania did not contain sufficient of these metals to develop other similar provinces.

With the appearance of the gabbro and norite in the Dundas district, sufficient nickel was included to give rise on differentiation to the fluids which formed the lodes of the Dundas nickel province. Elsewhere similar rocks were not accompanied by sufficient nickel to warrant the designation of the areas in which they occur as nickel provinces.

Acid Phase.—From the economic standpoint this is the most important phase of ore-deposition which has occurred in Tasmania. It has given rise to the following metalliferous provinces:—

The silver-lead provinces of Zeehan-Dundas, Mount Farrel, Mount Claude, and Magnet.

The zinc-lead-silver provinces of Read-Rosebery and Comstock.

All of the tin provinces of Tasmania.

The copper provinces of the West Coast Range, with the exception of those deposits included in the Epi-Cambro-Ordovician metallogenic epoch, but including the most important copper deposits of that region; also the copper provinces of Balfour and Scamander.

The iron province of Comstock.

The gold province of North-Eastern Tasmania.

It has been recognized for many years that a close genetic connection existed between the above deposits and the Epi-Silurian granite, but the explanation has been lacking of the actual details of the genetic relationship between the metalliferous provinces and the distribution of the consanguineous granite. This relationship has been worked out in detail by the writer in his work on the metallogenic epochs of Tasmania, referred to above.

Naturally, the details of that investigation cannot be presented here, but in essence the conclusion reached is that the present distribution of the ore deposits has been determined by the configuration of the granitic batholithic roof, which was itself controlled by the tectonics of the invaded sediments. This factor has been the minor control, and the major control has been effected by the original regional variation within the igneous magma. Minor control of the second order has been the result of purely local variations within the magma, and zonal precipitation has played a very subordinate part.

The mechanism by which the configuration of the batholithic roof has controlled the distribution of the ore deposits consisted of a relative transfer of the metals between the cupolas or upwardly projecting portions of the granite and the inter-cupola troughs. This transfer resulted in the concentration of the tin, tungsten, and bismuth, with boron, fluorine, etc., near the top of the cupolas, and a similar concentration of the lead, silver, gold, zinc, and copper, together with sulphur, etc., in the inter-cupola troughs. From these two types of concentration points they were ejected in successive stages, which are fully dealt with in the work above referred to.

With the exception, therefore, of the West Coast Range, Balfour, and Scamander, the metalliferous provinces enumerated above are in reality metallogenic provinces. The West Coast Range is a composite province of iron and copper, caused by the superposition of two metallogenic epochs on a persistent line of crustal weakness. The copper provinces of Balfour and Scamander are merely variants of the tin metallogenic provinces, and do not constitute definite copper provinces from the genetic standpoint.

(3) THE ECONOMIC SIGNIFICANCE OF THE GENESIS.

The above conclusions as to the distribution and genesis of the ore deposits of Tasmania possess a definite significance in regard to the future search for ore-bodies. Certain criteria can be enunciated which should guide future scientific prospecting in Tasmania.

Of the first order of magnitude is the criterion of the proximity, horizontally or vertically, of igneous batholiths of Epi-Cambro-Ordovician or Epi-Silurian age. Except for iron, and perhaps copper, search should be confined to the areas controlled by Epi-Silurian batholiths. Research must then be directed towards the delineation of the configuration of the batholithic roof. This will involve a close investigation of the tectonic geology.

The criterion of the second order of magnitude is based on the essential difference between the cupola and inter-cupola trough horizons. Tin, tungsten, and bismuth deposits need only be looked for where pronounced cupolas occur. The extent to which the cupola has been preserved from truncation by erosion is a valuable indication of the degree of probability of locating tin, tungsten, or bismuth deposits of economic value. An important conclusion in regard to the various tin provinces can thus be drawn. The North-Eastern tin province, as already indicated, consists for the greater part of alluvial deposits; at the same time it is evident, by reason of the extensive area of exposed granite, that truncation has taken place to a great degree. The second observation is the explanation of the first—namely, that the erosion of the tin lodes in the cupola which has been eroded has supplied the material of the existing alluvial deposits. Erosion has, in fact, proceeded so far that only the basal roots of the primary deposits remain. Consequently, this is not an area presenting much prospect of containing persistent primary tin lodes of economic value. This is confirmed by practical experience on tin lodes in this area. The Ben Lomond and St. Paul's Dome tin provinces, on the other hand, have suffered less truncation, and tin and wolfram lodes have been worked successfully. Search for similar lodes in these provinces is therefore justified.

The Dundas-Lindsay-Cleveland-Bischoff tin province consists of cupolas which still retain the greater part of their sedimentary cover, and very little truncation has occurred. Consequently, this province offers the best inducements for systematic search for primary tin lodes of all of the Tasmanian tin provinces. The Heemskirk cupola has been considerably truncated, but not sufficiently to remove all of the lodes, as some valuable deposits still remain. Nevertheless, there must naturally be far fewer lodes remaining than in the non-truncated cupolas.

The criterion of the third order of magnitude is provided by zones of orogenic crushing and faulting (as, for example, the overthrust of the West Coast Range).

Carrying the refinement to the fourth order of magnitude, search should be concentrated on those portions of the major movements which have taken place against solid buttresses, causing local crush zones—as, for example, that portion of the West Coast Range overthrust in which the Mount Lyell and North Lyell ore-bodies occur.

The criteria of lower orders of magnitude are provided by details of geologic composition and structure, which, however, need not be considered in such a short outline as this.

It becomes evident, therefore, that the ore deposits of Tasmania are confined to the western, north-western, and north-eastern portions of the State. With the exception of tin and tungsten, the valuable base metal deposits are confined to the north-western and western districts. Further than this the ore deposits are confined to circumscribed provinces which have been indicated above and depicted in the accompanying map. Ore-production of the future must depend on the intensive study of these provinces and the systematic investigation of neighbouring areas on the lines indicated above. The writer believes, however, that the ore deposits of economic importance are included in the provinces outlined above.

III.—COAL AND OIL-SHALE.

1.—Geographical Distribution.

COAL.

The coal deposits of Tasmania are for the most part confined to the east, south-east, and northern districts. The Preolenna and Barn Bluff–Pelion fields, in the north-western and western districts, are the only occurrences outside these main coal areas.

The map accompanying "Mineral Resources No. 7," recently issued by the Geological Survey of Tasmania, shows the various coal-fields and the actual coal-bearing areas, and it becomes clear, on studying this map, that the main coal resources are contained in the northern portion of the east coast area and in the portion of the south-eastern area bordering the D'Entrecasteaux Channel.

OIL-SHALE.

The tasmanite oil-shale occurs in four districts — namely, the Mersey Valley, Cam River (Oonah), Quamby Brook, and Karoola. The chief of these is the Mersey Valley, although the Cam River deposits have possibilities. These occurrences are shown in the map above mentioned.

Kerosene shale resembling torbanite, and a similar kerogenic coal termed pelionite, occur at Preolenna and Barn Bluff respectively.

It is thus seen that all of the oil-shale deposits are confined to the northern portion of Tasmania.

2.—Geology.

With the exception of the Preolenna, Mersey, Barn Bluff, Pelion, and Port Cygnet coal-fields, the coal deposits of Tasmania are of Trias-Jura age. The above exceptions are of Permo-Carboniferous age.

The Trias-Jura coal measures contain a maximum of eight seams, but in most coal-fields some of these are missing either by erosion or thinning out. The seams vary from a maximum of 16 ft. downwards.

The Permo-Carboniferous coal measures generally contain one seam, except at Preolenna, where four seams are locally developed. The thickness of these seams varies from 2 ft. downwards.

The coal measures of Tasmania are invariably horizontal, or only slightly inclined from the horizontal, but they are broken up by numerous tensional block faults and by constant transgressions of diabase. These structures have been investigated, and are described in the Coal Resources publication of the Geological Survey recently issued.*

3.—Potential Value.

With the exception of the coal at Preolenna, there is no high-grade coal in Tasmania. The Trias-Jura coal, while not high grade, is certainly of economic value. It varies slightly, but is of the same general character wherever it occurs. Although it possesses an ash content averaging about 22 %, it has a calorific value of 10,000 B.T.U. It therefore has a definite industrial value, particularly in view of the recent development in the use of powdered coal, for which purpose the Tasmanian coal is particularly suited. It is, however, a non-coking coal, and is useless for gas manufacture. The calculated reserve of this class of coal is 134,398,000 tons.

The Preolenna coals are rich in gas, and coke well. They are, however, rather high in sulphur. These coals, however, being all related to torbanite, will most probably be most economically utilized for distillation purposes. There is a calculated reserve of 5,000,000 tons of this class of coal, including the torbanite itself.

The actual value of the tasmanite oil-shales has yet to be ascertained. The technology of their distillation has not been sufficiently advanced to put the industry on a commercial basis. Much research work remains to be carried out on the distillation problem before satisfactory progress can be made.

IV.—RELATION OF THE ECONOMIC GEOLOGY TO THE HYDRO-ELECTRIC POWER RESOURCES.

In the map above mentioned the railways are indicated as well as the hydro-electric power plants. In addition, the area is shown within which the main hydro-electric power reserves are located.

Although in Tasmania distances cannot be regarded as great in actual mileage, yet the rugged topography makes the transportation problem an appreciable one. It is therefore a favourable combination of circumstances which has placed the most important metalliferous provinces adjacent or in close proximity to large potential hydro-electric power schemes. This more than compensates for the shortcomings of the coal and the geographical separation of the coal from the ore deposits.

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* Geol. Surv. Tas., Mineral Resources. No. 7.

The main railway communications are already constructed, with the exception of an outlet for the east coast coal-fields. The way is therefore open for future developments in mining and metallurgical operations.

As has been shown in this paper, Tasmania possesses important ore deposits. It is frequently claimed that this State possesses every known or important metal. This is misleading, and future prospects cannot safely be built up on that basis. Certainly Tasmania possesses a wonderful variety of mineral deposits, but when we come to get down to solid, practical facts it is found that there are appreciable deficiencies. Thus, Tasmania possesses, as far as present knowledge goes, no workable deposits of chromium, vanadium, molybdenum, cobalt, or aluminium, very little arensic, practically no precious stones, no graphite of value, and no radium minerals. There is only one nickel province, and that is of limited potentialities ; but, in view of its being the only reserve of payable nickel in the Commonwealth, it assumes a special importance from the national aspect.

It is particularly unfortunate that there are no aluminium ores, as, with cheap electrical energy, a valuable industry would quickly arise.

Nevertheless, Tasmania possesses deposits of tin, copper, zinc, lead, silver, gold, osmiridium, iron, coal, and oil-shale, which, apart from cement, ceramic, paint, and similar raw materials, provide a very solid basis for important industrial developments.

HIGH-GRADE ZINC IN INDUSTRY.

BY HARRY HEY.

FOR some years past it has been usual to specify high-grade copper for use for many purposes, but the practice of specifying zinc of a high degree of purity for certain purposes is of comparatively recent origin. It is a practice, however, which is extending as the effect of the impurities in zinc on products manufactured therefrom becomes widely recognized.

The notes hereunder are intended to briefly survey the principal uses to which high-grade zinc is being put, and, wherever possible, to indicate the effect of the impurities normally present in the lower grades of zinc.

In the United States zinc is graded into five divisions, known respectively as "High-grade," "Intermediate," "Brass Special," "Selected," and "Prime Western." The maximum amounts of impurities allowed in the various grades are indicated in Table I., which is compiled from data published by the American Society of Testing Materials :—

TABLE I.

	Lead	Iron	Cadmium	Alu-minium	Total Lead, Iron, & Cadmium
	%	%	%	%	%
High-grade	0.07	0.03	0.07	..	0.10
Intermediate	0.20	0.03	0.50	..	0.50
Brass special	0.60	0.03	0.50	..	1.00
Selected	0.80	0.04	0.75	..	1.25
Prime Western ..	1.60	0.08	..	..	..

In addition, it is specified in all cases that the slabs shall be reasonably free from surface corrosion or adhering foreign matter.

Although zinc used in Europe has in many cases been sold on analysis, it is only recently that any attempt has been made to grade the metal. The London Metal Exchange published quotations for what is known as "G.O.B." spelter—in other words, as "good ordinary brand," which includes brands containing impurities in amount as high as the Prime Western grade of the United States. This does not, of course, imply that all the zinc used in Europe is of no higher grade than G.O.B. A considerable quantity of electrolytic zinc is now consumed for special work, and commands an appreciable premium; but prior to 1914 the amount of high-grade zinc utilized there, apart from that used in the fabrication of cartridge brass, was small.

The world's production of high-grade metal during 1913 did not much exceed 20,000 tons, but during recent years the quantity produced has greatly increased, due largely to the war experience with the pure metal in making brass for munitions and to the successful exploitation of the electrolytic method of recovering zinc. The production during 1922 reached approximately 119,000 tons, of which 106,000 tons was electrolytic zinc, as compared with a total zinc production of all grades of primary zinc of 716,000 tons.

Apart from the plants capable of producing high-grade distilled zinc, there are at present electrolytic zinc plants in existence with an annual capacity of at least 130,000 tons of zinc, all of which can be marketed as high-grade metal.

Before discussing the various uses to which the high-grade metal is put, a survey of the impurities present in G.O.B. spelter, together with the effect such impurities have on the properties of the metal itself, is advisable.

The most common impurities present in zinc are lead, cadmium, and iron, although arsenic sometimes occurs in small quantities.

Lead forms alloys with zinc, and various observers have indicated that when a mixture of zinc and lead is subjected to liquation the lead content of the zinc layer cannot be commercially reduced below 1 %, and Spring-Romanoff indicate a figure as high as 1.5 % as being the solubility of lead in zinc under commercial conditions at a temperature slightly above the melting point of zinc. The presence of lead in zinc tends to promote cracking in zinc castings. It renders it weak, as evidenced by the liability of galvanized coatings containing excessive lead to crack and peel.

Cadmium in zinc behaves in a manner different from lead. Zinc and cadmium are mutually soluble when molten, but are mutually insoluble when a mixture of these two metals is solidified. Thus, when a molten mixture of cadmium and zinc is allowed to solidify, metallic cadmium and metallic zinc separate, and exist in the solid as such.

Recent work by Pierce (*Amer. Inst. Min. and Met.*, 1923) indicates that, by long annealing, a cast mixture of zinc with small amounts of cadmium will become homogeneous. The effect of even traces of cadmium is apparent, particularly in zinc castings. Cadmium imparts brittleness to zinc, and makes it hot short. Rigg and Morse (*Amer. Inst. of Metals*, ix., 49) describe a test which illustrates the weakening effect of cadmium. A metal mould similar in shape to a flattened dumb-bell was provided. It consisted of two discs 2 in. in diameter joined by a neck $\frac{1}{4}$ in. x $\frac{1}{4}$ in. x 2 in. Pure zinc and zinc containing varying amounts of cadmium were cast in this mould. Sound castings of the high-grade metal could be obtained at any temperature, but with zinc containing 0.1 % Cd it was impossible to produce a sound casting even when the metal was poured near its freezing point, the castings showing cracks in the shank. With 0.05 % cadmium perfect castings could only be obtained by casting at the lowest possible temperature.

Table II., quoted by Stone (*Engineering and Mining Journal*, vol. cii., p. 909), indicates the effect of cadmium on the compressibility and brittleness of zinc.

TABLE II.

EFFECT OF CADMIUM ON HARDNESS OF ZINC.

% Cd.	Load for 10 % Compression lb. per sq. in.	Relative.
0.00	16,246	100
0.02	18,660	115
0.05	20,080	124
0.08	20,950	129
0.20	26,870	165
0.25	28,270	174
0.27	28,700	177

Iron is present in most spelters in varying amount, but even the poorest G.O.B. quality should not contain more than 0.1 % iron, although published analyses indicate that occasionally European spelters contain iron in excess of this quantity. Iron forms definite compounds with zinc, the most common of which is that having a composition approximating $FeZn_7$. This apparently dissolves zinc giving a composition equivalent to $FeZn_{11}$. When more than 0.1 % iron is present in zinc the metal has a distinct greyish appearance. and frequently shows black specks throughout the crystals. The effect of iron is to increase the brittleness of the zinc and reduce its malleability.

Arsenic has a hardening effect, and tends, in excess of 0.01 %, to make the zinc very brittle.

ALLOYS.

One of the most important uses of high-grade zinc is in the manufacture of alloys. Alloys containing zinc are legion, and they range from what are known as " white brasses," which contain up to 85 % zinc and anti-friction metals, through the various yellow and red brasses to the bronzes. The aluminium zinc alloys are a comparatively recent development, and the uses therefor are increasing year by year.

The various forms of brasses are better known than other classes of zinc alloys, and it has long been recognized that, in order to produce a brass which has to withstand spinning and deep drawing, it is necessary to rigidly exclude the impurities which are commonly present in the lower qualities of zinc. These fine brasses are subjected to excessive strain during working, and a brass containing lead and iron particularly is weak, and will not stand up to the work that is necessary to obtain the desired shape. Moreover, the effect of any overstrain may not become apparent until the manufactured brass has been stored for some time.

Typical of this class of fine brass is what is known as cartridge brass, The term " cartridge brass " usually refers to pure zinc-copper alloys containing from 65 % to 75 % copper. The desired properties in an alloy for cartridge brass are such that strength has to be combined with malleability and ductility to facilitate working, and at the same time the finished product is required to withstand the severe shock to which it is subjected in the breech of the gun or rifle.

As illustrating the changes through which the alloy passes from the ingot to the finished strip, the following figures given by Brownsdon (*J. S. C. I.*, 1917, p. 795) are of interest :—

TABLE III.

	Yield Point.	Tensile Strength.	Elongation.	Hardness. Brinell.
	Tons.	Tons.	%	
Ingot (chill cast) ..	6	16	60–70	60
Strip hard rolled ..	over 60	30–40	10-15	150–200
Finishing strip annealed at 650° C. ..	6	20–23	65–75	60

The tensile strength figure given above for the finished strip represents British practice.

The changes indicated are severe, and can only be met by using the purest of metals for the alloy. Lead reduces the strength of brass, as it separates between the crystal faces of the brass ; thus the alloy is rendered weak at these points. The lead limit in cartridge brass is usually specified at 0.025 %.

Iron renders brass brittle, and the specification for fine brass limits the iron content to a very low figure. Small amounts of iron can be counteracted by subjecting the brass to more frequent annealing during rolling. In certain cases the iron content of zinc has been limited to as low a figure as 0.015 % ; electrolytic zinc can easily meet this standard. Iron also tends to produce an alloy which develops fine hair cracks during drawing or spinning. These fine cracks are particularly objectionable when it is desired to plate the resulting article, as the plating solution finds its way into any crevices, and local corrosion is set up, with the result that coloured salts of copper are formed, which stain the finished article and render it unsightly.

A few words about the effect of cadmium in brass. This impurity has been the subject of controversy by different interested parties. One faction maintains that cadmium is in part, at any rate, responsible for season cracking, or, as it has recently been called, with corrosion cracking in brass, and the other faction argues that there is no evidence that this is so, and further, owing to its volatility, very little cadmium remains in the alloy after pouring. The question of corrosion cracking has been fully covered by a number of papers in recent years, and a good deal of attention has been given to the effect of overstrain during cold rolling, and not a great deal as to the effect of impurities. It is fully agreed by most brass manufacturers, as stated above, that iron and lead are deleterious, and from what we know ot cadmium one would expect that segregation would occur during crystallization. Exactly how much cadmium is necessary for the segregation to be serious enough for the formation of high cadmium alloy or cadmium metal between the crystal faces of the brass the present writer does not know. The fact, however, that pure zinc is now available at a price very little higher than G.O.B. spelter should persuade the brass manufacturer to use only the highest-grade metal, particularly as the question of cadmium is still the subject of controversy. According to Stone, brass containing cadmium is more sensitive to heat treatment; consequently one would expect it to be more susceptible to overstrain during working. In passing, it should also be mentioned that the determination of cadmium in brass is not an easy matter. Its determination in zinc is tedious, and requires considerable skill and experience; but when copper is present in addition the difficulties are greatly increased.

The manufacture of condenser tubes is another branch which calls for a type of brass which should be made only from the purest metals. This, of course, means that under present conditions high-grade zinc should be used.

The corrosion of condenser tubes has been the subject of prolonged research, and much has been written regarding the causes thereof and the methods of preventing same. The variables bearing on this question are numerous, and in a general paper such as this it is not intended to survey in any way the work undertaken in this direction. Sufficient is it to say that, as all experience has shown that the presence ot impurities in brass facilitates cracking during working, lead, iron, and cadmium should be eliminated as far as possible from an alloy which is to be subjected to the strain during working and the conditions in service that condenser tubes have to undergo.

The alloys of zinc with aluminium with or without small amounte ot copper are most interesting. Such alloys are now largely used for die-casting. They possess relatively low melting points, ars very fluid when molten, and are not " hot short," and thus enable castings free from fracture to be easily produced.

The use of zinc for alloys of this type is steadily extending, particularly as the cause of the failures experienced during the early attempts to use such alloys becomes apparent.

When aluminium zinc alloys were first used, various objections were raised owing to the frequency with which the alloys failed in service, particularly when they were exposed to a warm, moist atmosphere. The castings swelled and distortion was apparent, accompanied by severe cracks. In some cases the castings disintegrated. This phenomenon did not occur in every casting, and it was apparent that there was some variation between the composition of the good and bad castings or that the method of casting differed.

Rawdon (*Trans. Amer. Elec. Chem. Soc.*, 1921) ascribes these failings to intercrystalline oxidation, and Bauer and Peirce (*Amer. Inst. Min. and Met. Eng.*, lxviii., 1923) have recently investigated the nature of this intercrystalline corrosion or oxidation in as far as the zinc rich alloys are concerned. The work of these investigators indicates clearly that sound castings can be obtained with either the binary alloys of zinc-aluminium or the ternary alloys of zinc, aluminium, and copper, provided that high-grade alloying metals are used. The results obtained show that in some cases a few hundredths of 1 per cent. of an impurity accelerate the oxidation when present in an alloy exposed to wet steam for a relatively short period.

These results coincide with Rosenhain's experience with the aluminium-zinc alloys rich in aluminium. The wet steam test, which has been used by various investigators, is drastic, but it enables results to be obtained quickly, and the results obtained on various samples are certainly relative to one another.

Lead, which is the most frequent impurity in zinc, is by far the most powerful accelerator. Cadmium also is very active, but iron is apparently not so detrimental. Copper, on the other hand, when present between 0.5 % and 5 %, exerts a retarding action.

Apart from the failure of the castings, the impurities present in the lower grade of castings affect the bending properties of the alloys.

According to Bauer and Peirce, the most satisfactory zinc-rich alloy is that containing zinc 92 %, aluminium 5 %, and copper 3 %, produced from high-grade metals. This alloy has a tensile strength of 40,700 lb. and has excellent casting and bending properties.

The zinc aluminium alloys rich in aluminium are used extensively where castings possessing strength combined with lightness are required. The strongest aluminium rich alloy is that containing 30 % zinc, which has a tensile strength of 18 tons, although the most popular alloy for small castings, such as gear boxes, crank cases, etc., contains 15 % zinc.

An alloy used for aircraft purposes, and which is a serious rival of duralumin, contains aluminium 72 %, zinc 25 %, and copper 2 %. It possesses strength, and after hot rolling has an elongation of 21 % on 1 in.

In addition to the zinc-aluminium and zinc-aluminium-copper alloys, die-casting alloys rich in zinc and containing tin, copper and aluminium are extensively used. Table IV. gives assays of alloys typical of this class.

TABLE IV.

Zinc.	Tin.	Copper.	Aluminium.
87.5	8.0	4	0.5
73.75	14.75	5.25	6.25
46.2	30.8	20.4	2.6
86.0	8.0	5.5	0.5

GALVANIZING.

The coating of iron and steel with zinc, in order to produce an article possessing the strength of steel and at the same time resistant to atmospheric corrosion, has been practised economically since Crawford commenced to exploit the hot dipping method. According to Sang (*Rev. de Met.*, 1912), iron was first coated with zinc by Malorin in 1741, but the value of a zinc coating as applied to iron was not recognized until nearly a century later. The discovery of the effect of corrugating on the strength of thin steel sheets, in 1851, however, was largely responsible for the expansion of the galvanized sheet industry. During the 19th century the production of galvanized sheets, wire, etc., extended rapidly, and at the present time approximately 50 % of the world's output of zinc is used in galvanizing. The galvanizing industry is thus of paramount importance to the zinc producer.

The term "galvanizing" is a misnomer, as electricity plays no part whatever in the application of the zinc coating to iron and steel excepting in the case of the comparatively small amount of iron coated by electro-galvanizing. The question of electro-galvanizing or of sherardizing will not be discussed in this paper.

Zinc is an excellent protective coating provided the iron surface is completely covered, and galvanized ware is used in many forms, but the bulk of galvanizing is carried out on sheets, wire, and wire-netting.

For some years now the galvanizer of wire has used the purest metal procurable, and he has particularly specified that cadmium and iron should be very low—in fact, the United States wire galvanizers have paid a substantial premium in order to obtain metal free from these impurities. Cadmium is more objectionable in the galvanization of wire than in any other industry in which zinc is used, excepting in the manufacture of slush castings, which will be referred to later. As small an amount as 0.05 % of cadmium tends to make a zinc coating brittle, and brittleness is particularly undesirable in the zinc coating of telephone and telegraph wires, which are subjected to severe bending and twisting during erecting. It is the practice for the post-office officials in various countries to subject any galvanized wire, before purchase, to the test known as the "lineman's joint," which is a severe bending similar to that which the wire has to undergo in service, and the supplier is forced to the use of zinc which contains little cadmium.

The presence of lead tends to produce a weak coating with a tendency to peel, but it does not appear to be as detrimental as is cadmium, provided it is not present in excess. Excessive lead also renders zinc coatings more liable to corrosion, particularly when exposed to soft waters. On the other hand the presence of lead in amounts of from 0.5 % to 0.75 % appears to increase the fluidity of zinc, and thus enables a thinner coating to be obtained.

Iron increases the viscosity of molten zinc, and the coating produced therefrom is usually uneven and brittle.

With high-grade zinc with or without small amounts of lead, and provided the bath is not too hot, excellent coatings of great flexibility can be obtained. The author has seen samples of telephone wire which have been galvanized with pure zinc, and which have been twisted under tension with the two ends rigid until the steel has fractured. The zinc coating followed the distortion of the steel and was still intact. There was nothing special about these samples, which were taken from standard 8-gauge wires, and were representative of material being produced in quantities at Newcastle, New South Wales.

Whereas in galvanzing wire the desideratum is the production of a coating which will be elastic and resistant, the production of galvanized sheets calls for something more, which, from the sheet galvanizer's standpoint, is equally important. Everybody is familiar with the "spangled" effect of zinc-coated sheets. The best-known makers of galvanized iron grade their sheets on the appearance; but, in explanation of this, the same makers claim that the public judge the value of the sheet by the brightness and the form the hoar-frost-like spangles take. This spangled appearance can be easily obtained by a skilled galvanizer from a good grade retort spelter relatively free from iron, but containing lead, and to which a small amount of tin has been added. Strange to say, however, on first attempting to galvanize with a bath of high-grade zinc the appearance of the sheet was anything but what was desired. Following on this experience, the Electrolytic Zinc Company in 1921 carried out a series of experiments in a small-scale mechanical pot which was a replica of the galvanizing equipment used on the commercial scale. The size of the pot was such that sheets 3 ft. 9 in. long by 9 in. wide could be conveniently galvanized.

Experiments were undertaken to determine the most satisfactory conditions for galvanizing when using pure zinc and zinc to which varying quantities of lead and tin were added, both singly and in conjunction.

The investigation was interesting in that it was definitely indicated that if an artistic appearance is desired it is necessary to have a certain amount of lead present in the molten zinc.

The effect of lead is remarkable, and the best results were obtained when this metal was present in quantities between 0.6 % and 0.9 %, together with 0.3 % tin. With lead below 0.6 % the background was rather dull, and the spangles were not well defined, whereas the spangles produced with lead present in excess of 0.9 % were "spiky" —in other words, were too pointed.

This is one of the very few cases where the presence of lead in zinc is found to be an advantage.

The next problem was to apply these experimental results on a practical scale. At first sight it appears simple, but the mere fact of adding the necessary amount of lead to the molten zinc contained in the galvanizing bath does not ensure that the added metal will be evenly distributed throughout the mass. This method of doctoring zinc was tried, but the results were not encouraging.

The method which has finally been adopted is to operate the bath of zinc over a layer of from 6 in. to 8 in. of molten lead. When melting in a fresh pot the lead and zinc are well stirred by means of dry compressed air, so as to incorporate the necessary amount of lead through the zinc, and thus start away with a bath of the desired composition. When galvanizing sheets, the usual practice is to add slab zinc in order to replace the metal removed from the bath on the sheets, and thus maintain the metal level in the pot at the desired height. Solid zinc, being heavier than molten zinc, sinks down from the molten zinc and melts in contact with the lead below.

In some cases the rate of addition of slab zinc is too rapid to allow sufficient lead to be absorbed, and it is necessary, periodically, to blow the zinc wich compressed air. This can conveniently be carried out at the week-ends, when the pot is being drossed.

There does not appear to be any reliable experimental data as to the effect lead has on the life of a galvanized sheet when exposed to normal atmospheric conditions. It is interesting, however, to consider this question on theoretical grounds.

The galvanized coating is really a thin casting of zinc bonded to the iron or steel sheet by zinc-iron alloy of more or less thickness, depending on the amount of iron in the zinc bath, and more particularly on the temperature of the bath. The behaviour of the zinc during cooling is identical with an ordinary zinc casting containing equivalent amounts of lead and tin. Such a mixture is homogeneous when molten, but becomes heterogeneous when it solidifies. It therefore follows that the chemical composition of the galvanized coating produced from zinc containing lead and tin is not regular over the sheet. The spangles probably differ in composition from that portion of the coating represented by the bright background. Thus, on immersion in an electrolyte, a couple will be formed and corrosion will be concentrated at certain parts of the surface—in other words, the spangles would be dissolved before the bright background, or *vice versa.*

Recent work by Hannah and Rhead (*Inst. of Metals*, Sept., 1923) confirms the above hypothesis. Dilute hydrochloric acid attacks the zinc coating of a spangled sheet irregularly, and also certain portions of the surface dissolve more readily than others when the sheet is subjected to the salt spray test. It is to be noted, however, that immersion in dilute hydrochloric acid, or even applying the salt spray test, does not altogether parallel the conditions under which galvanized iron is used, particularly in relation to couple action, but the results are interesting, nevertheless, and are to some extent indicative.

The use of high-grade zinc in galvanizing has extended during the past two years, and considerable quantities have been used in Australia and elsewhere for this purpose. The galvanizer is realizing that he can depend upon obtaining regular results when using a metal of regular composition, provided he maintains an even bath temperature, because the fluidity at any given temperature is constant.

No attempt is made in this paper to describe the process of galvanizing or the equipment used therefor. An excellent paper by Sang (*Rev. de Met.*, 1912), and also Flanders's book, " Galvanizing and Tinning," give detailed descriptions.

SHEET ZINC.

The sheet zinc industry is an important one, and, although it is difficult to state the world's consumption of zinc as sheets, the amount used for this purpose in the United States during 1922 totalled 53,500 tons, which includes the rolled zinc used for boiler protection. The consumption of high-grade metal in the sheet industry is increasing as the properties of zinc free from impurities become better known. For some time it was contended that zinc free from lead could not be rolled. During recent years, however, this has been proved to be a fallacy, and it is now generally recognized that for most classes of sheet the pure metal is preferable. It is difficult to obtain a hard sheet from very pure zinc, but, on the other hand, sheets possessing superior bending properties can be produced. Thus it is that the prejudice that metal workers had against zinc sheets in general is gradually disappearing, as experience with pure zinc in sheet form becomes more general. Practically all metals when added to zinc in small quantities (such as a few tenths of 1 %) increase the hardness, but at the same time reduce its resistance to bending. The impurities normally in zinc tend to harden the sheet, and, if present in excess, render it brittle. Cadmium is particularly prone to do this. Lead renders the sheet weak.

The resistance to bending, which is a measure of the value of sheet zinc to the metal worker, does not appear to be a function of the hardness of the sheet, but it depends rather on the purity of the metal from which this sheet is rolled. Mathewson, Trewin, and Finkeldey (*Amer. Inst. of Min. and Met.*, 1920, p. 358) report the results of experiments carried out on strip rolled zinc of varying hardness. Four samples were obtained which were called hard-rolled, medium hard-rolled, soft-rolled, and dead soft-rolled respectively. The first three were rolled from what might be termed an excellent quality of common spelter assaying—Pb, 0.26 % ; Cd, 0.08 % ; Fe, 0.02 %—whereas the No. 4 was formed of Horsehead quality, which is equivalent to the best grades of electrolytic zinc. The hardness was regulated by varying the speed with which the ingot was reduced to the required thickness (0.010 in.) Cold bending tests were carried out on these four samples. The size of the test sheets was 1½ in. x 1 in , and the test was carried out by bending the specimen backwards and forwards. A bend through 180° was counted as one complete bend, and a bend backwards through 180° another bend, and so on. Tests were carried out both with and across the grain. A number of tests were reported on unannealed samples and on samples which had been annealed at various temperatures between 50° C. and 300° C. On referring to Table V., which is compiled from Mathewson, Frewin, and Finkeldey's figures, it will be seen that annealing at 225° C. improved the bending properties of the sheets rolled from the lower-grade zinc considerably, but with one exception the results obtained were not as good as the unannealed figures of the sheet produced from the pure metal. The effect on the soft rolled sheet was sufficient to keep it well ahead of the impure sheets, but was not quite as marked as with the latter. The figures quoted are selected from the series, and are the maximum number of bends obtained on each sheet.

The results are reported as the number of bends necessary before the zinc shows the first sign of failure.

TABLE V.

BENDING TESTS AND SCHLERESCOPE HARDNESS FIGURES ON UNANNEALED AND ANNEALED ZINC STRIP.

(*a*) *Unannealed.*

	Hard rolled.	Medium.	Soft.	Dead soft. (high grade)
Hardness	29	24	15.5	15.0
Number of bends—				
Across grain ..	1.5	4.0	9.0	19.5
With grain	0.5	1.0	4.5	9.5

(*b*) *Annealed at* 225° C. *for one hour.*

	Hard rolled.	Medium.	Soft.	Dead soft. (high grade)
Hardness	14.0 ·	13.0	14.5	13.5
Number of bends—				
Across grain ..	21.5	15.0	(5 hr.) 13.5	25.5
With grain .. .	7.5	5.5	7.5	15.5

It will be noted that there was only a slight difference between the unannealed hardness of the softest of the sheets produced from the zinc containing impurities and the sheet produced from the pure metal.

It is obvious from the figures given in Table V. that a sheet rolled from high-grade zinc has properties which render it available for a variety of uses. The range of sheet zinc products is surprising to the uninitiated. Its properties are such that it can be subjected to deep drawing, spinning, stamping, etc., without showing any tendency to fracture. It is non-staining, and thus has an advantage over copper or brass for many purposes, as, for example, the manufacturer of corsets is now using eyelets and protecting tips for the corset steels from zinc instead of from brass.

It is unnecessary to specify the great variety of uses to which sheet zinc is put, but before leaving this phase it is worthy of mention that the dry battery manufacturers in Australia—and, it is understood, in other countries also—are insisting that the zinc they use in the fabrication of the zinc shell of the battery shall be pure, and they specially require the sheet to be free from lead.

Considerable quantities of zinc sheets are converted into zinc shavings for use in the precipitation of gold from cyanide solutions. It has been maintained by many writers that such zinc should contain lead to be efficient, but commercial tests carried out by certain Rand mines proved that sheets rolled from electrolytic zinc were at least equal to sheets in general use in South Africa. It is general Rand practice to immerse the shavings in lead acetate solution prior to their use for gold precipitation. This practice is independent of the composition of the zinc.

The indications are that high-grade zinc possesses advantages over spelter when used for boiler preservation. The so-called zinc boiler blocks are a rolled product, and are usually manufactured by breaking down an ingot $1\frac{1}{2}$ in. in thickness to a plate, say, $\frac{1}{2}$ in. to 1 in. thick, which is then sawn into sizes suitable for bolting into the inside of the boiler shell. Everybody is familiar with the function of these blocks. In the presence of an electrolyte (the feed water functioning in this regard), the zinc and the sheet of the boiler shell form a couple, and as the zinc is electro-negative to iron, any corrosive action possessed by the feed water is concentrated on the zinc instead of being distributed over the boiler shell. As the zinc contained in the blocks is consumed, the blocks are replaced.

It was anticipated that pure zinc would show some advantage over zinc containing impurities owing to the absence of segregated metals or alloys, as, for instance, the lead-rich zinc, which separates when zinc solidifies. Tests on a laboratory scale were undertaken by the Electrolytic Zinc Company, and the results therefrom gave very definite indications that the pure zinc corroded at a slower rate than did the blocks rolled from spelter, when both were heated in contact with feed water. Further tests on a steamer confirmed in some degree the preliminary tests, but further work is necessary before any definite conclusion can be arrived at.

For the manufacture of lithographic sheets, which are rolled sheets, high-grade zinc is now usually specified. The presence of impurities in a sheet rolled from impure zinc results in uneven etching owing to the heterogeneous nature of the surface of the plate.

Slush Castings.

The production of slush castings is not a very extensive industry, and it is confined largely to the United States. It is a process principally used for the manufacture of ornamental articles which are required to be hollow and of light weight. The metal to be cast is poured into moulds until full, and the mould is quickly emptied. A thin wall remains as a shell casting, similar in shape to the inside of the mould. The latter, which is usually made of bronze, machined to the desired pattern, is in two halves, so held that it can be quickly taken apart for the removal of the casting.

For slush castings the metal must be such that it will melt at a low temperature into a mobile liquid, and will solidify into a rigid casting which, under tension, retains a surface free from cracks. Given such a metal, sound castings of intricate pattern can be produced. It is essential, however, that the surface of the castings should be free from even hair cracks, as it is usual to electroplate or stain the casting before it is ready for use.

Zinc, relatively free from impurities, is highly suitable for this work. Its melting point is low; it is mobile a few degrees above its melting point; it is not hot short, and consequently does not develop cracks during cooling; and, finally, it is easily electroplated with any of the metals or alloys likely to be used for that purpose.

The presence of even small amounts of the impurities present in common spelter produces faulty castings. This effect of impurities has been thoroughly investigated by Rigg and Morse (*Amer. Inst. of Metals*, ix., 29), and their results form the basis of this section of the paper. Details of the equipment used, together with a number of photographs, are given in the paper referred to.

It was found that cadmium had the most deleterious effect. Even when this impurity was present to the extent of only 0.05 % it was impossible to produce sound castings with temperatures as low as 425° C. Even with this small quantity of cadmium the metal was "hot short." Furthermore, by increasing the weight beyond the commercial limit no casting was obtained which was strong enough to withstand the strains set up during solidification and subsequent cooling.

Lead, as an impurity, although not so harmful as cadmium, reduces the tensile strength of the metal. When the zinc solidifies, minute globules of lead segregate and remain liquid after the zinc has solidified. The presence of this liquid lead renders the casting weak, and cracking results. With a lead content below 0.2 %, and when the metal is cast at relatively low temperature, good results can be obtained, but owing to the lower temperature the castings are heavier, and thus zinc containing lead is uneconomical. With increasing lead the results become less satisfactory. It is to be noted, however, that the surface of slush castings produced from metal containing up to 0.6 % lead is satisfactory.

With zinc containing iron the results are unreliable and erratic. In general, iron increases the viscosity of zinc, and the weight increases accordingly. An increase in temperature of the metal does not appear to counteract this tendency for heavier castings. It is submitted that iron segregates as $FeZn_7$, and concentrates at a point which will be weak and unable to withstand the cooling strain.

The above results are particularly interesting because they indicate the effect of impurities on the behaviour of zinc during casting.

ZINC OXIDE.

A considerable amount of metallic zinc is used in the manufacture of zinc oxide. All grades of zinc are used for this purpose, but if an oxide free from lead and cadmium is required it is necessary to rigidly exclude lead and cadmium from the metal used as raw material. The major portion of the higher grades of zinc oxide is used in the manufacture of rubber goods and in the preparation of fine enamels.

Zinc oxide increases the strength of rubber. Good plantation rubber vulcanized with sulphur, and without the addition of reinforcing pigment has a breaking strain of approximately 1500 lb. per sq. in. If a mixture of 50 % zinc oxide with 50 % of rubber and sulphur combined is vulcanized, a sheet is produced which has a breaking strain of approximately 2900 lb. per sq. in. The extensibility of the rubber is reduced, but at the same time it retains its spring and resiliency. If lead is present in the zinc oxide, lead sulphide is formed during vulcanization, and, as it is dark in colour, it stains the finished rubber.

Lead compounds in zinc oxide are also undesirable when the pigment is used for fine enamels. Such compounds appear to combine with the gums contained in the varnish vehicle, and the resulting compounds tend to thicken the enamel.

Any cadmium contained in metal used for zinc oxide manufacture volatilizes with the zinc and forms cadmium oxide, which is brown in colour and possesses powerful staining properties. A relatively small amount of cadmium oxide imparts a distinct yellowish tint to zinc oxide.

The above summary of the uses to which high-grade zinc is put by no means exhausts the subject, but it briefly covers the major industries in which it is used. The regular composition of a metal which is required to comply with a standard of 99.9 % purity makes such a metal of value to all users of zinc. The recognition of this is becoming more apparent, and its use for general purposes is thus expanding in such industries where, at first sight, a pure metal does not appear essential.

PROCEEDINGS AUS. I.M.M. (INC.)] [N.S., No. 52, 1923.

THE HANDLING OF MATERIALS AT THE RISDON WORKS OF THE ELECTROLYTIC ZINC CO. OF AUSTRALASIA LTD.

BY A. W. COOK, B.Sc.

THIS paper deals with the handling of the various materials used in the 100-ton zinc plant at Risdon, and, in view of the magnitude of the works, the configuration of the ground covered by these works, the variety of the materials used, and the large quantities, a very attractive and interesting subject is opened.

The works are situated on the right bank of the River Derwent, about four miles above Hobart, the capital city of Tasmania.

The Risdon works cover a space of 149 acres, the area being 3800 yards long by 2800 yards wide, and the elevations vary from 6 ft. above high-water mark at the wharf and foreshore to 163 ft. above high-water mark at the power station, these being the limiting points as regards elevation which are considered in this paper. Throughout the paper elevations are given in feet above high-water mark.

Fig. 1 shows a general view of the works taken from the opposite bank of the River Derwent. The wharf is clearly shown, with s.s. *Kekerangu* loading residue; behind the wharf is seen the inclined conveyor leading to the calcine bin; further up the hill is the leaching division. No. 1 haulage runs to the left of this up to a point on the left of the power station, the large reinforced concrete building at the top of the hill. Below this is the cell room, the largest building on the plant. The roasting division, not very clearly seen, lies at the foot of the main stack, and the casting division lies to the left of this, between the leaching division and the cell room.

A large wharf has been constructed alongside the works. This wharf is 880 ft. long and 60 ft. wide. It is capable of berthing large overseas steamers, as at low tide there is a depth of 30 ft. of water. The rise and fall of the tide at the wharf is 4 ft. 6 in., and the river is navigable at all states of the tide. Just above the company's wharf

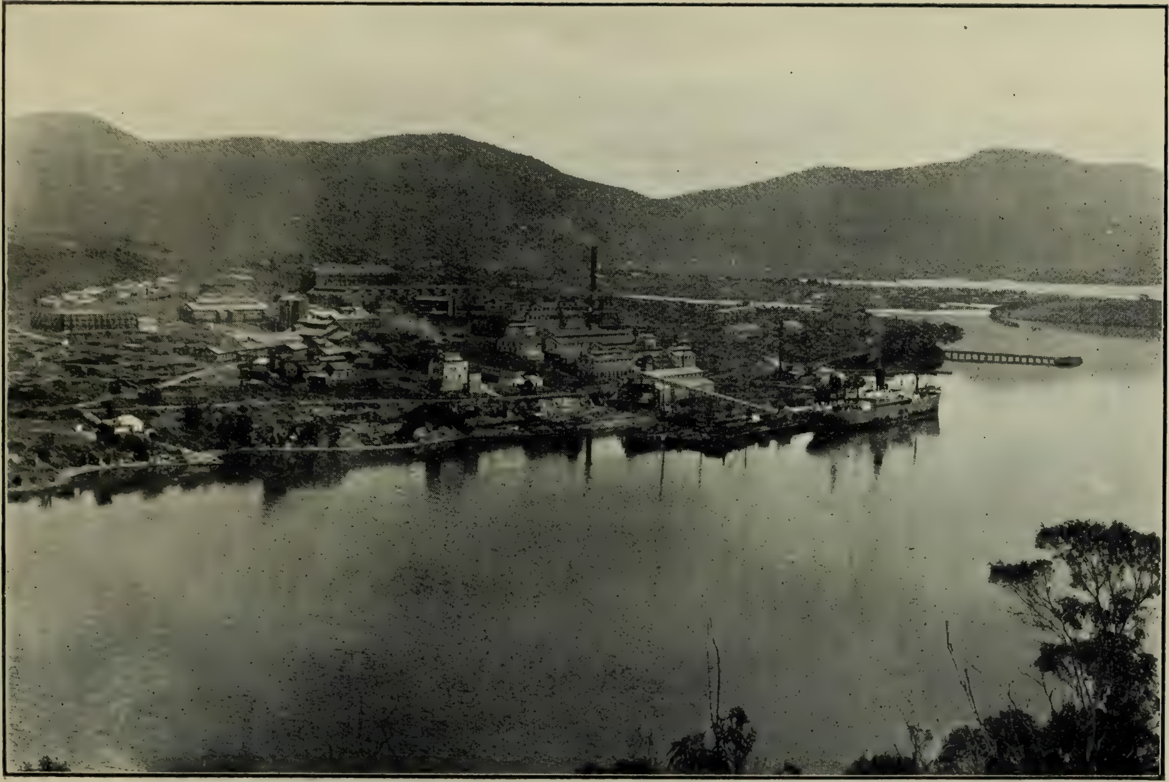

Fig. 1

is a natural basin, in which large steamers can turn in safety without the assistance of tugs. In fact, in no part of the port of Hobart is a tug necessary for berthing purposes.

For the convenience of steamers arriving and departing from Risdon wharf at night time, arrangements are under consideration by the Hobart Marine Board, in conjunction with the company, for lighting the passage up the river where necessary.

The materials to be considered are as follows:—

(*a*) Calcine, which may be classed as the raw material supplied to the works, and from which the zinc is extracted.
(*b*) Coal.
(*c*) Lime-rock or limestone.
(*d*) Cathode zinc, the metallic form in which zinc appears in the electrolytic cells.
(*e*) Slab zinc—the marketable form of zinc.
(*f*) Residue from the leaching plant, which is shipped to Port Pirie for further treatment.

The handling of these materials will be considered as fully as possible, and the photographs are attached to the paper for the illustration of several features.

(*a*) HANDLING OF CALCINE.

The calcine used is brought from the mainland of Australia in steamers carrying up to 5000 tons.

The calcine is lifted out of the ship alongside the wharf by means of self-acting grabs, each taking a load of three tons. These grabs are lifted by electrically-operated cranes of the portal type, supplied by Messrs. Babcock and Wilcox, of Renfrew, Scotland. These cranes, of which there are two, each have a lifting capacity of 7 tons at a radius of 50 ft., and have hoisting, luffing, slewing, and travelling gear, each gear having its own motor. The hoisting motor is 85 b.h.p., operated by a contactor panel, and the maximum load of 7 tons can be raised at the rate of 120 ft. per min. A self-setting limit-switch is provided on the hoisting gear to prevent over-winding. The lowering control is by means of a solenoid brake fitted with a hand release, and also by a foot brake. In case the rate of lowering becomes too rapid, a centrifugal brake comes into operation when the speed of the motor reaches 750 r.p.m. The jib of the crane is balanced, and, owing to this arrangement, the full load can be luffed at the rate of 200 ft. per min. by a 5 b.h.p. motor in either direction. A solenoid brake is fitted to the luffing gear, and an over-luffing switch comes into operation at maximum and minimum radius. The slewing gear is operated by a 15 b.h.p. motor and controlled by a foot brake. The travelling gear is operated by a 15 b.h.p. motor, and the cranes can be moved along the wharf at the rate of 50 ft. per min. The driver's cab is 34 ft. above the deck of the wharf, and a clear view is obtained of all operations. All controls are effected from the cab, and all motors can be cut out and brakes put on in an emergency by one push-button. Power feeders are laid under the wharf, connecting boxes being fitted at suitable intervals. Flexible armoured cables form the connection between these boxes and the under-carriage of the cranes. The leads to the motors pass up the king-post from a plug on the under-carriage, and the current is collected by brushes from rings whose centre coincides with the slewing axis, a similar connection being made from these rings to the travelling motor, which is fixed half-way up the substructure.

These cranes run on 100-lb. steel rails laid flush with the decking, and can operate over the full length of the wharf.

With these cranes and the conveying gear (to be described later) it is possible to unload 4500 tons of calcine in about 32 hours. The unloading plant can be put into operation a few minutes after the steamer is moored and the hatches opened.

The grabs in use are self-acting, of the Preistman type, which close upon their load as soon as hoisting commences, and, when the load is lowered, the grab discharges itself at the first hoist after landing on the wharf or hopper.

Fig. 2 shows s.s. *Era* unloading calcine at the company's wharf. The cranes are well shown, also the hoppers into which the calcine is discharged and the first (No. 1) conveyor.

The cranes disharge their load into travelling hoppers, which can be moved to any point on the conveyor below that is convenient to the hatches of the steamer alongside. These hoppers are of steel, and are 15 ft. long by 15 ft. wide, having a capacity of 50 tons if required. On the top of these hoppers are heavy timber baulks, which form a solid grid on which the cranes land the grabs, and in so doing they are discharged into the hoppers below. With careful handling of the cranes there is very little shock on these hoppers. At the bottom of these hoppers, and part of the hoppers themselves, are revolving rollers 12 in. diam., having large longitudinal grooves. These rollers run at 20 r.p.m., and give a continuous feed to the conveyor belt below. In case any large lumps occur in the calcine, and this is frequent, balanced doors are provided below, which open automatically and let the lumps pass through on to the belt. This method answers well unless the calcine is very damp; then trouble occurs through bridging of the calcine inside the hopper. This is overcome by frequent poking by hand.

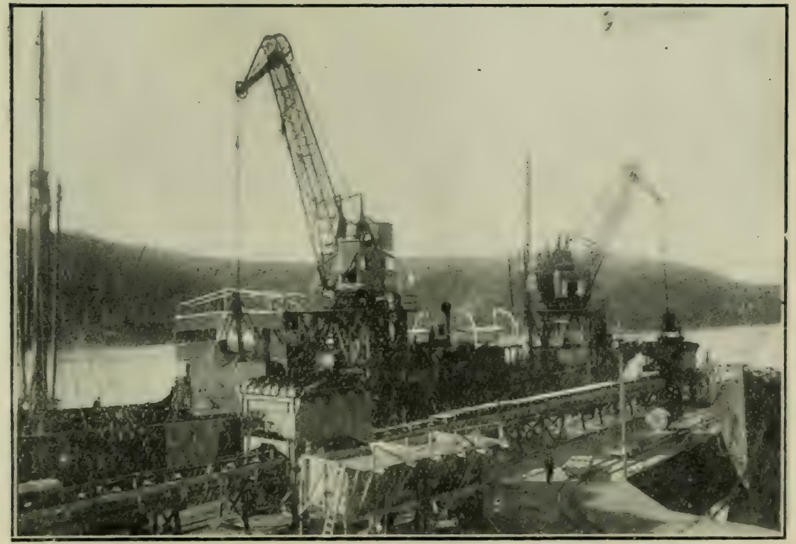

FIG. 2.

S.S. "ERA" DISCHARGING CALCINE.

Underneath these hoppers is fixed a belt conveyor at such a height as gives headroom below across the wharf. This conveyor (No. 1) has a rubber-covered belt 24 in. wide which runs on 20° troughing idlers, the rubber on the working side of the belt having a thickness of $\frac{1}{8}$ in. The length of the conveyor is 280 ft. centres, its speed 338 ft. per min., being driven by a 25 b.h.p. motor at the head end. The joint in the belt is a half lap one, 24 in. long; this has proved very satisfactory, and has been adopted on all the conveyors. At the discharge end this conveyor is raised to enable it to deliver on to an inclined conveyor leading up to the calcine bin on the foreshore.

This inclined conveyor (No. 2) is of the same design, having a 24-in. rubber belt, its length is 300 ft. centres, and speed 467 ft. per min. It runs up a grade of 1 in 6, and is driven by a 40 b.h.p. motor at the head end. This conveyor is carried on a timber structure resting on piles driven into the river bed between the wharf and the foreshore. It discharges through a chute on to a horizontal conveyor (No. 2A) of similar design, length 210 ft. centres, having a 24-in. belt running at 500 ft. per min., and is driven by a 15 b.h.p. motor at the tail end. The drive of this conveyor has not been as satisfactory as the other drives owing to the weight of calcine on the belt reducing the ratio of tensions. This conveyor is carried on steel

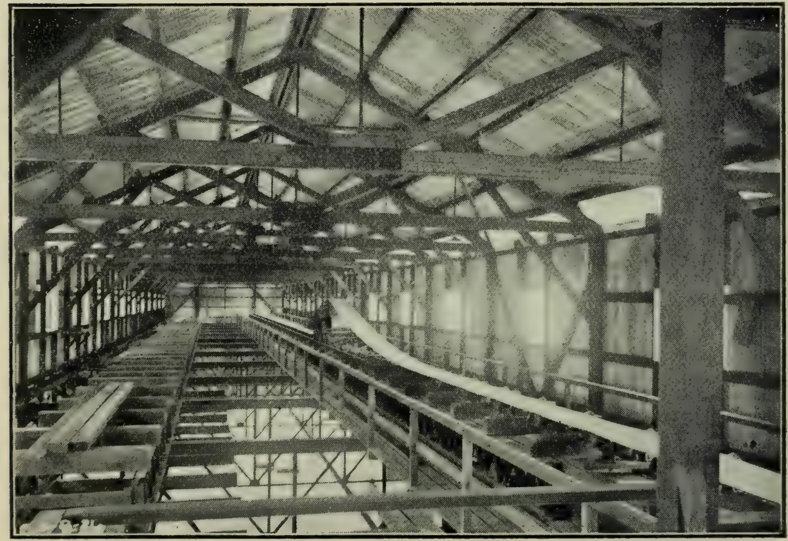

Fig. 3.
No. 2A belt conveyor in calcine bin.

built-up girders resting on the roof trusses of the calcine bin. A tripper is fitted to the belt so that the calcine can be discharged to any part of the bin. The distance from the top of the belt to the floor of the bin is 45 ft. Provision has been made for two conveyors in this bin, but it has not been found necessary to instal both.

Fig. 3 shows No. 2A conveyor and tripper, also the interior of the calcine bin during construction.

The calcine bin is situated on the foreshore alongside the Risdon road, and is of reinforced concrete construction as regards the floor and the lower part of the walls; it has a steel roof, and is housed in with corrugated iron. The length at present is 190 ft., width 80 ft., height to under side of conveyor girders 40 ft. Its capacity is 18,000 tons of calcine. On the river side of the bin are a series of timber chutes through which the calcine is charged into trucks, and arrangements are also made for a conveyor to be installed alongside.

The equipment of the calcine bin consists of a Thew shovel, electrically driven, capable of lifting a ton of calcine. This shovel works in conjunction with a portable belt conveyor, and by means of this combination calcine can be picked up from any part of the bin and charged into the trucks alongside. The Thew shovel has only one motor, 20 b.h.p., and when well handled can easily deal with double the amount of calcine required. Fig. 4 shows this shovel without the cab, and shows all the gear, also the reinforced wall of the calcine bin.

Calcine is taken from the bin to the roasting division by means of trucks up the main haulage. This is of 2-ft. gauge, and runs up to an elevation of 160 ft., the maximum grade being 1 in 5. This haulage runs under the Risdon road and the main works road at the 76-ft. level. It is connected to the wharf, lime-rock storage, reserve coal storage, main works road, roasting division, and a number of other points. The distance up the haulage and down to the roasting division is 700 yards, and the return trip takes 9 minutes, a rake of trucks being in waiting at either end. The haulage winch is driven by an 86 b.h.p. motor, and there are 1650 ft. of 13/16-in. diam. steel rope on the drum. From the turnout leading to the roasting division to that division there is a fall of 42 ft., and the trucks are lowered by the winch under gravity, and, although the destination and most of the trackage is out of sight, there is very accurate control, and accidents are very rare. A system of signalling by electric bell pushes is in operation throughout the haulage.

The trucks are taken up in rakes of ten, making a load of 9 tons of calcine. The trucks are of the side-tipping dump pattern, the axles running in roller bearings. To give greater freedom on curves, one wheel on each axle is free, the other being forced on. The free wheels are placed on opposite sides of the truck. There are draw-bar connections through the undercarriages. These trucks give very good service.

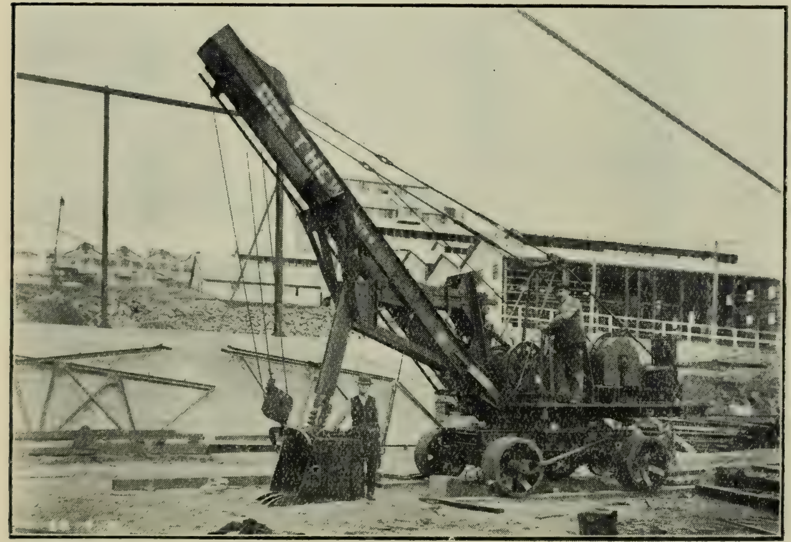

Fig. 4.
Electrically-operated "Thew" shovel.

On arrival at the roasting division the trucks pass over a weighbridge and are tipped into an elevator boot through a wood grizzly, to extract lumps. The calcine is then raised by an elevator of the bucket type running at 440 ft. per min. and passed through a trommel on to a belt conveyor leading to the bins of the roasting furnaces. The screening from the trommel passes over a magnetic pulley to extract pieces of tramp iron, and is afterwards ground in a Parkinson disintegrator

The fine in the bins is passed into the four hearths of the Leggo roasting furnaces by means of ribbon conveyors and screw feeds, each feed having four double screws 7¼ in. diam. at 6¾ in. pitch, operated by adjustable ratchets. The two lower screws have a shield over them to prevent the calcine becoming too dense. The top hearth can be charged direct from the conveyors if desired.

The calcine is passed along the hearths of the roasting furnaces by means of revolving rabble arms rotating about an axis vertical to the hearth, adjacent arms revolving in opposite directions, and thus move the material down the gradual slope of the furnaces.

At the discharge end of the hearths the last rabble arm passes the roasted calcine into the discharge chute, and the chutes from each of the four hearths meet in one common chute; this terminated on a plate attached to the furnace push-conveyor below. As this conveyor moves to and fro, a predetermined amount of red-hot calcine is allowed to rill down an inclined plate into the trough of the push-conveyor between two adjacent blades, thus giving a dustless discharge. This is rather important, as hot calcine is very mobile and liable to dust.

The furnace push-conveyors, 18 in. wide, one to each furnace and running parallel with it, are set at the same incline as the furnaces —viz., 1 : 12. Each one discharges on to a plate moving with the main push-conveyors through an inclined hopper. These hoppers are set at such an angle as will prevent the calcine sliding down, and thus forces it to form a rill; to make this doubly sure, small angles are rivetted across the bottoms of the hoppers to further retard its movement. This special precaution and the one described in the preceding paragraph were adopted to prevent dusting, and they have proved successful.

The main push conveyors into which these hoppers discharge are 36 in. wide and 150 ft. long, making 30 strokes per min.; they are inclined upwards to the wetting-down plate at a grade of 1 : 20. These conveyors are balanced by counter-weights, and are very effective both in handling the quantity required and in cooling the calcine.

The main push-conveyors discharge the material, which is wetted down and delivered through a trommel on to an inclined belt conveyor (No. 8).

This conveyor, having an 18-in. rubber belt, runs on 20° troughing idlers and has a length of 220-ft. centres ; the speed is 210 ft. per min. It runs up a grade of 15°, being driven by a 10 b.h.p. motor at a point about half-way up. This conveyor is so arranged that it can be discharged either on to another conveyor (No. 9) or into a storage bin. This bin is provided so that in the event of a stoppage in the roasting division stock can be drawn through a "chinaman" and transferred by trucks to No. 8 conveyor to ensure continuity of operations in the leaching division.

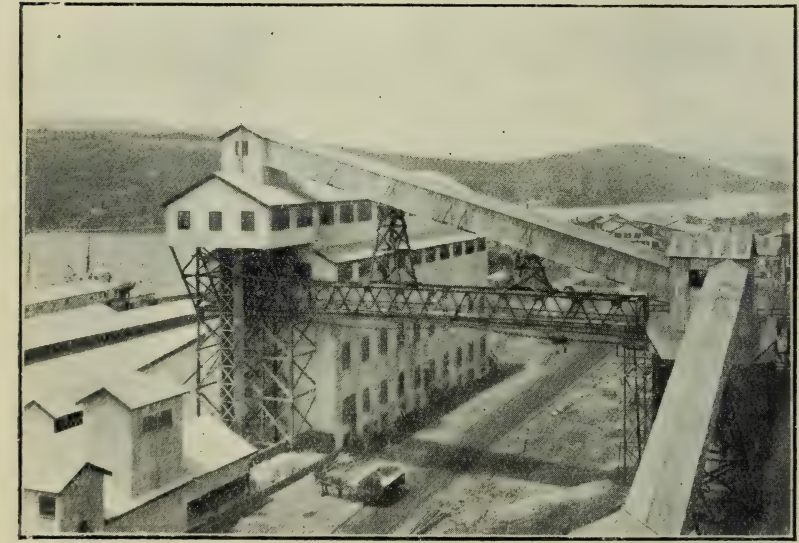

Fig. 5.

Inclined conveyors from roasting division to leaching division.

No. 9 conveyor takes up its load from No. 8 conveyor and passes it up to the storage bin at the top of the leaching division. It goes up an incline of $16\frac{1}{2}$°, its length being 140 ft. centres, width of belt 18 in., speed 150 ft. per min., and motor 10 b.h.p. This conveyor is carried on timber trestles supported on steel girders 45 ft. above the main works road at the 76-ft. level. This bridge also carries a number of pipes. Nos. 8 and 9 conveyors are fitted with a device for cleaning the under side of the belts. This is a revolving drum having four rubber vanes; these vanes frequently strike the under side of the belt and keep it clean.

Fig. 5 shows No. 8 and No. 9 conveyors, the storage bin, the pipe bridge, and the leaching building, also the main works road at the 76-ft. level and railway track.

The discharge from No. 9 conveyor falls into the 50-ton storage bin at 170-ft. level, 12 ft. above top floor of the leaching division. Calcine is drawn from this bin by six chutes, three on each side, fitted with arc gates, and at present discharge into standard dump trucks; but arrangements are in hand for the installation of two large trucks, one for each side, propelled by cable power, each carrying a load of 4½ tons. The dump trucks, after passing over a weighbridge, are tipped into leach plant bins. The large cable-operated trucks when in operation will discharge through bottom drop-doors.

At this point the handling of calcine as a solid substance ceases. It will be of interest here to note the manner in which the solutions are handled.

The natural slope of the ground on which the leaching division is built enables a gravity flow to be utilized very largely in passing the solutions through the division. The pure solution is lifted from the lower end of the leaching division at 37-ft. level to the pure solution tanks supplying the cell division at 175-ft. level by centrifugal pumps. The return solution from the cell division after electrolysis runs back by gravity to the leaching division for re-use.

(*b*) HANDLING OF COAL.

The amount of coal used per week is 440 tons, and it is received in railway trucks on the 76-ft. level from the Tasmanian Government Railways, whose lines run into the works at this level.

Coal is also drawn when required from the reserve coal storage below the main works road. This is handled by the main haulage, switched on to the road level, and transferred to the coal hopper by means of dump trucks. The coal is discharged into the coal hopper and elevated to a point above the level of the roasting division coal-bin, where it is discharged into a trommel. The fine passes into the coal-bin on a flat belt conveyor, and is then drawn as required by the mechanical stokers of the roasting furnaces. The nuts either fall back into a crusher, and are again elevated, or they are transferred to the melting furnaces. The mechanical stokers are of the type with a travelling grate, the movement of which can be varied as required by a small hand-wheel. The stokers can be removed bodily from the furnace fire-boxes when required.

As mentioned above, some of the nuts from the screen are transferred by trucks on an elevated trackway to the coal-bins of the zinc melting furnaces. These bins discharge their coal, in the case of No. 1 furnace, into the hopper of the mechanical stoker, also of the under-feed type, and in the case of No. 2 furnace, which is hand-fired and principalty used as a stand-by, on to the firing floor in a convenient position.

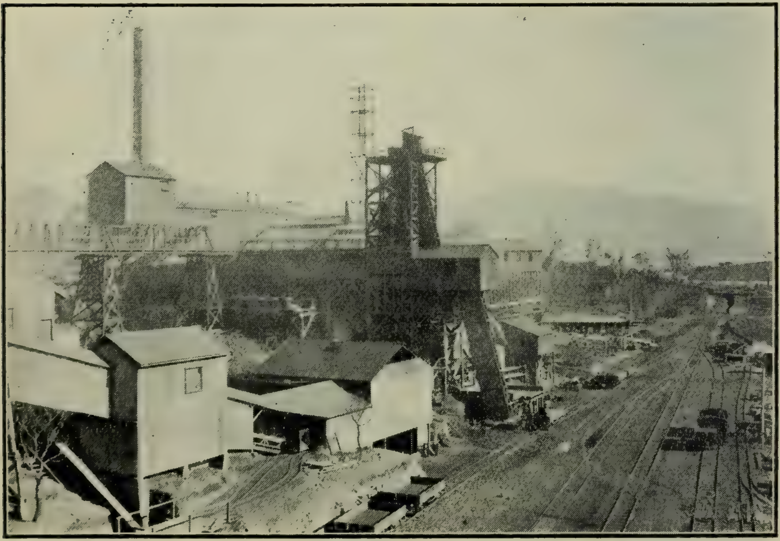

Fig. 6.
Roasting division showing coal-handling plant.

The coal for the Stirling steam-boilers and driers in the leaching division is trucked from the railway siding to that particular coal-bin on an elevated trackway across No. 3 haulage.

Fig. 6 shows the coal elevators and bins of the roasting division, also the elevated trackway to the melting furnaces.

(*c*) HANDLING OF LIME-ROCK.

Lime-rock, or limestone, to the amount of 10 tons per day is used in the leaching division, and is obtained from quarries in Ida Bay, in the D'Entrecasteaux Channel, about 60 miles south of Risdon. It is brought to the lime-rock jetty, about 50 yds. south of the main wharf, by river steamers, and is stored on the foreshore.

As the quantity used is small compared with that of the calcine, no elaborate plant for rapid discharge of the small steamer is provided. Trucks are run along the jetty to the steamer, the hood is lifted off, and lowered into the hold ; when full it is raised and replaced on the under-carriage, the truck is then run back, and its contents dumped on the storage heap. All lifting is done by the ship's tackle.

From the storage dump the lime-rock is trucked up the main haulage, switched off at the main works road at the 76-ft. level, and then transferred to the lime-rock section at the up-river end of the leaching division. This section is shown at the left-hand bottom corner of Fig. 5. In this section the lime-rock is crushed and ground into a slime by jaw-crusher rolls and tube mill, the latter being in closed circuit with a classifier ; it is then elevated by a self-discharging bucket into a storage bin at 142-ft. level in the leaching division. This lift is also shown on Fig. 5.

(*d*) HANDLING OF CATHODE ZINC.

Cathode zinc is the product of the electrolytic cells in the cell room. The zinc deposit is stripped off in a plate form by hand tools. The cathode zinc is then loaded on trucks ; each cathode truck carries the product of one cell—about 1 ton of zinc. The trucks run on 24-in. trackage and are passed out of the cell room on to a platform, over a weighbridge, and on to an elevator, which lowers them to the level of the drying tunnel. This elevator is of the Johns and Waygood patttern, having a lifting capacity of 50 cwt. ; it is driven by a 15 b.h.p. motor through worm gear, and operates over a distance of 32 ft.—from the cell room floor-level to the level of the lower trackage. Two trucks go down the elevator at one time. When the trucks, with their load of zinc cathodes, reach the level of the drying tunnel they are removed from the elevator and passed through the tunnel, which has a slight grade downhill. After leaving the tunnel the trucks pass along the trackage over the furnaces; the zinc cathodes are then charged into the furnaces by hand through hoppers in the top. The empty trucks are then moved to the elevator on a return trackage and lifted to the cell room level for recharging.

(*e*) HANDLING OF SLAB ZINC.

Slab zinc is the marketable form of zinc, and consists of a slab 18 in. x 9 in. x 1½ in. thick, weighing about 56 lb. The amount of slab zinc to be handled is at the rate of 125 tons per day, and special plant was devised for it.

Consideration of this material will be commenced at the tap holes of the melting furnaces, where it appears, of course, in a molten form. The metal is tapped out into luted cast-iron ladles having a capacity of 5 cwt. each. These are carried by hand operated 1-ton hoists suspended from trolleys running on 8-in. x 4-in. steel joists. These runways extend to the mould benches, two in number, and are so equipped with a switch that the desired liberty of action as regards destination can be attained. The 5 cwt. of molten zinc is hand-poured into ten cast-iron moulds mounted on steel benches, each bench having three sets of ten moulds, thus giving 60 moulds altogether. When the metal has set, these ten moulds are tipped in groups of five, and the slabs fall reverse side uppermost on to a chain conveyor. A conveyor is fitted under each of the two benches. The conveyors are made up of steel links, 12-in. centres, and by suitable clutch gear either conveyor can be moved at a speed of 15 ft. per min. Their length is 42 ft., centre to centre of sprockets, and they extend a few feet beyond the end of the bench. These conveyors discharge into a steel chute leading to a gravity roller conveyor, and, as the chute is rather long, the slabs acquire a high velocity; this is retarded by steel gates at their lower ends. By this means the slabs, on landing on the roller conveyor, have not sufficient momentum to cause them to rebound. The roller conveyor is set at right angles to the chutes, and is fixed at such a slope as will enable the slabs to acquire sufficient velocity to clear the end of the chute before the next slab slides down. This conveyor, after passing the second chute, is curved at a radius of 8 ft. through a quarter of a circle and then terminates in a straight portion. As the slabs run on this portion they are led into a definite position by side guides so that they can be taken up by the spacer. The office of the spacer is to take the slabs off the roller conveyor singly and deposit them on the main slab conveyor leading to the wharf. This spacer is made up of two steel discs 10 in. apart, mounted on a revolving shaft; 2-in. angles are rivetted to the inner faces to form six radial ledges—on these the slabs are carried. The roller conveyor terminates between these faces, and as the spacer revolves the end slab is lifted up; the next one slides forward on the rollers into its place, and is taken up by the following pair of spacer angles. The spacer turns the slab over, and its fall from the back of one angle to the face of the one in front is retarded by a weighted lever. To protect the driving gear a copper shear-pin is fitted in the driving wheel, so that if a slab fouls the spacer the extra load causes this pin to shear, and the spacer stops; but as this pin is quickly replaced, very little delay is experienced, and the spacer, when re-started, picks up the waiting slabs one at a time and passes them on to the slab conveyor. The

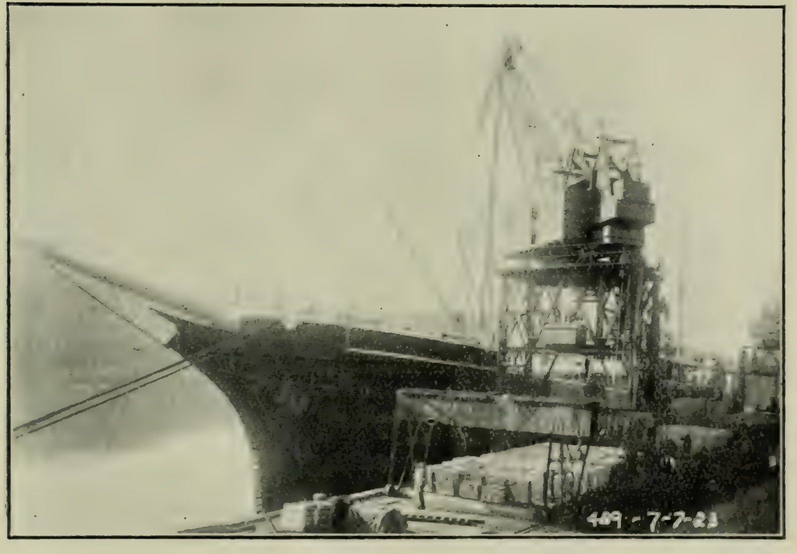

Fig. 7.
S.S. "Eastern" loading zinc slabs.

spacer and the main slab conveyor are so timed that when one slab is ready to slide off the angle shelves at 30° inclination, the spurs on the main conveyor, which at this point has an inclination of 35°, are there to receive it.

The main slab conveyor is an endless chain 6 in. wide, having steel links 2 in. x $\frac{3}{8}$ in., with 12-in. centres, and spurs are fitted every 24 in. It has a total length of 520 ft., centre to centre of the sprocket wheels. It travels from an elevation of 95 ft. at the head end to an elevation of 8 ft. at the tail end. The maximum grade is 35° ; this is the first flight below the spacer. This part of the conveyor is shown on the left-hand bottom corner of Fig. 6. The conveyor passes under the main works road and railway through a tunnel, and down the hillside under the Risdon road, terminating on the foreshore. The conveyor moves at the rate of 11 ft. per min., and the slabs have time to cool sufficiently for stencilling and handling on the wharf. It is driven by a motor of 15 b.h.p., which also drives the conveyors

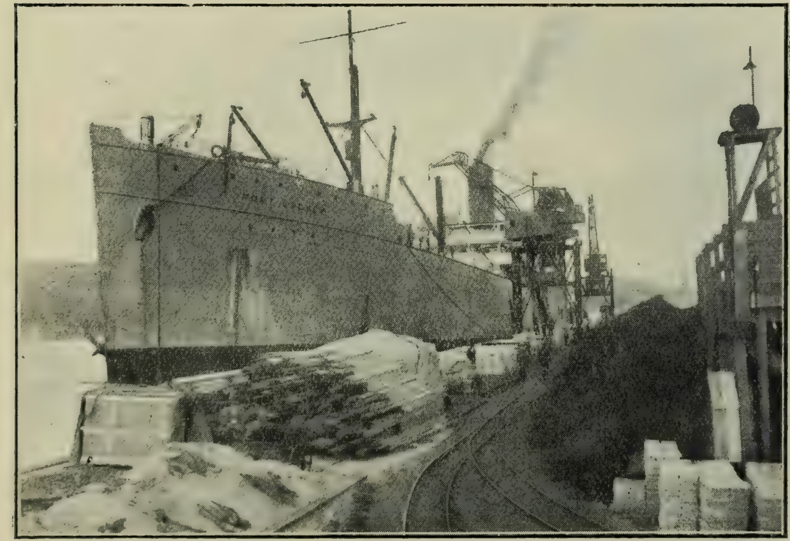

FIG. 8.

S.S. "PORT SYDNEY" 9128 TONS, LOADING ZINC FOR UNITED KINGDOM.

under the mould bench and the spacer. This can be stopped if necessary from any point by pulling a wire rope which extends the full length of the conveyor.

The slabs are taken off the conveyor by hand and placed on flat-topped trucks on skids in regular piles, their position being located by a stacking guide. This arrangement enables a block of 60 slabs —*i.e.*, a truck-load, approximately 30 cwt. in weight—to be lifted off the truck bodily, a special cradle being used for this purpose. The stacks of slabs are lifted off on the wharf by means of a travelling gantry 28 ft. long, 12 ft. 6 in. high, and deposited on skids lying on the decking of the wharf. This gantry and the blocks of slabs are shown on Fig. 7, which shows s.s. *Eastern* (3586 tons).

The wharf cranes, when loading zinc slabs, are fitted with special cradles to lift these blocks of slabs, and by means of a crossbar two cradles are lifted at once—a load of 3 tons of zinc. The cranes lower these cradles into the steamer's hold, where the load is released by catches. The slabs are then stacked in the hold by hand. This reduces the handling of the slabs to a minimum.

The two wharf cranes can load zinc slabs into a ship at the rate of 100 tons per hour.

Fig. 8 shows s.s. *Port Sydney* (9218 tons) loading zinc at Risdon wharf for shipment to the United Kingdom. This steamer is the largest that has come up to the wharf.

(*f*) HANDLING OF RESIDUES.

The residue from the leaching division is shipped from Risdon to the mainland for the extraction of lead and silver. The calcine steamers usually take this residue as a return load instead of going back in ballast.

The residue is in a finely-divided form, and is easily handled. It is the cake from the filters; this is a chocolate-coloured mud, carrying 30 % moisture, and is dropped from the filter into a large wood hopper. From this it is forced by screw feeds into two inclined revolving driers. Passing through these, it is discharged at 12 % moisture in a coarse form on to a belt conveyor, the belt of which is specially prepared to carry a hot substance. This conveyor passes under the fire-boxes of the driers, having a belt 18 in. wide and a length of 50 ft. between centres. This in turn discharges on another belt conveyor 18 in. wide, having a length of 120 ft. between centres. It discharges into a bin alongside the Risdon road. Below this bin is a track-way 24 in gauge, and the residue is drawn from the bin and trucked to a dump on the shore side of the up-river end of the wharf. The wharf cranes load this material into the steamers with the self-acting grabs, and can handle 250 tons per hour.

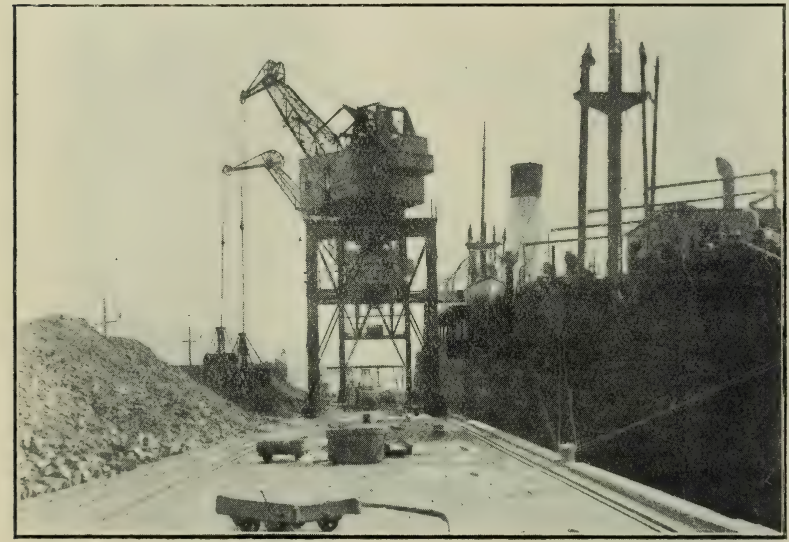

FIG. 9.
S.S. "ERA" LOADING RESIDUE.

Fig 9 shows s.s. *Era* loading residue from the dump on the wharf.

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THE PRINCIPLES AND APPLICATIONS OF THERMO-ELECTRIC PYROMETRY.

BY LEONARD B. HANEY.

INTRODUCTORY.

THERE are few industrial and metallurgical processes which do not require the application of heat energy, and the modern demand for increased working efficiency has resulted in greater attention being paid to temperature control as a means of ensuring fuel economy and more efficient plant operation.

In general, the accuracy required is not very great except in laboratory work, and few industrial operations are controlled to within less than 5° C., while in many cases greater limits are allowable.

Almost every thermal property of matter has been suggested as a basis for the construction of pyrometers, but the thermo-electric type finds by far the widest application, and hence this paper will deal only with the principles, construction, and uses of this type of instrument.

THERMO-ELECTRIC PYROMETRY.

In the year 1821 Seebeck discovered that if the ends of two dissimilar metal wires were joined and one junction heated, a current flowed in the circuit, and the E.M.F. (electro-motive force) set up could be measured by inserting a galvanometer at the cold junction. The phenomenon was not thoroughly investigated for many years, but, as the demand for high-temperature thermometers increased, several investigators, of whom Le Chatelier was one of the pioneers, successfully endeavoured to utilize the principle in pyrometric construction.

The couples which gave the most satisfactory results were discovered, improvements were made in material and design, and the present-day type of instrument gradually evolved.

The apparatus consists of three main parts, viz. :—

1. The thermo-couple itself of two dissimilar metal or alloy wires welded together at one end.
2. The leads which connect the terminals of the thermo-couple with the measuring instrument.

(3) The instrument for measuring the E.M.F. set up, which may be either a galvanometer or a potentiometer.

Figs. 1 and 2 diagrammatically represent the apparatus, the former showing the galvanometer and the latter the potentiometer type.

In Fig. 1, A is the thermo-couple, for example, of iron and constantan, B the leads of copper wire, and C the galvanometer (a millivoltmeter). At the hot junction an E.M.F is set up which causes a current to flow from constantan to iron and thence through the instrument. Fig. 2 shows in outline the potentiometer type. An E.M.F. from a constant source—*e.g.*, a battery, B, is made to oppose the E.M.F. of the couple, T, and the two are balanced by the aid of a bridge, a highly-sensitive galvanometer, G, being used as indicator. The bridge wire is calibrated beforehand, and the temperature is calculated from the length of bridge wire used.

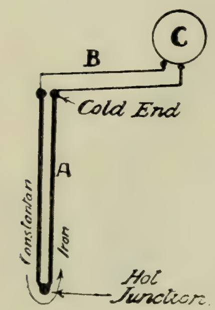

FIG. 1.

SIMPLE DIAGRAM OF GALVANOMETER METHOD.

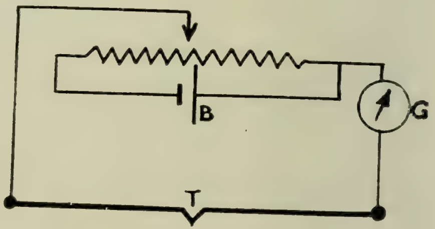

FIG. 2.

SIMPLE DIAGRAM OF POTENTIOMETER METHOD.

The chief advantage of the potentiometer method is that there are no resistance troubles, since no current flows. The chief disadvantage is that the readings are not automatic.

Recently an apparatus has been devised which operates on a combination of the galvanometer and potentiometer methods. It has proved very satisfactory, but has not yet received extensive application.

THERMO-COUPLES.

The E.M.F. developed by a couple is dependent upon the difference in temperature between the hot and cold junctions, and arises from two main sources. The first and most important is the

E.M.F. developed at the hot junction of the two dissimilar metals, called the Peltier E.M.F.; the second is an E.M.F. set up in each wire alone. In any homogeneous wire whose ends are at different temperatures, an E.M.F. exists. Though small, it is sufficiently large to have an appreciable effect on the total E.M.F. of the couple, which is, of course, the algebraic sum of the individual E.M.F's.

Couples may be divided into two classes—precious metal and base metal—according to the materials of which they are composed. The former consist almost entirely of platinum and its alloys, while the latter are composed of various base metals and alloys, the most important of which are :—Chromel (nickel-chromium)—Alumel (nickel-aluminium); Iron-Constantan (nickel-copper); Chromel-Nickel alloy; Constantan-Copper.

PROPERTIES OF COUPLES.

The chief properties which are desirable may be summarized as follows :—

(1) Ability to resist oxidation and corrosion under operating conditions.

(2) The development of a reasonably large E.M.F.

(3) The relation between E.M.F. and temperature to be such that the E.M.F. increases continuously with increase in temperature over the whole range employed.

(4) Permanency and consistency.

(5) Reproducibility and ease of manufacture.

(6) Cheapness consistent with satisfaction in other properties.

These properties may now be dealt with in greater detail.

(1) *Oxidation and Corrosion* troubles are usually encountered in base-metal couples. Pyrometer couples are subjected to great heat, often under severe operating conditions, and may be exposed to oxidizing gases, acid fumes, metallic vapours, molten metals, etc. Oxidation causes the most trouble, and great care must be taken to provide proper insulation and protection for couples to prevent failure. When couples are oxidized appreciably they change in resistance and the E.M.F. they produce, so that there is a tendency for eddy currents to be produced, and these effects reduce the accuracy of the instrument.

(2) *The Development of Relatively Large E.M.F.*—The highest E.M.F. developed by any base-metal couple does not exceed 50-60

millivolts at 1000° C., while that of platinum-rhodium couples is only approximately 14 m.v. at 1400° C. From these figures it is obvious that, even with base-metal couples, detecting instruments of great delicacy must be employed. The higher the E.M.F. developed the more robust may be the construction of the instrument, and this fact is important in plant work.

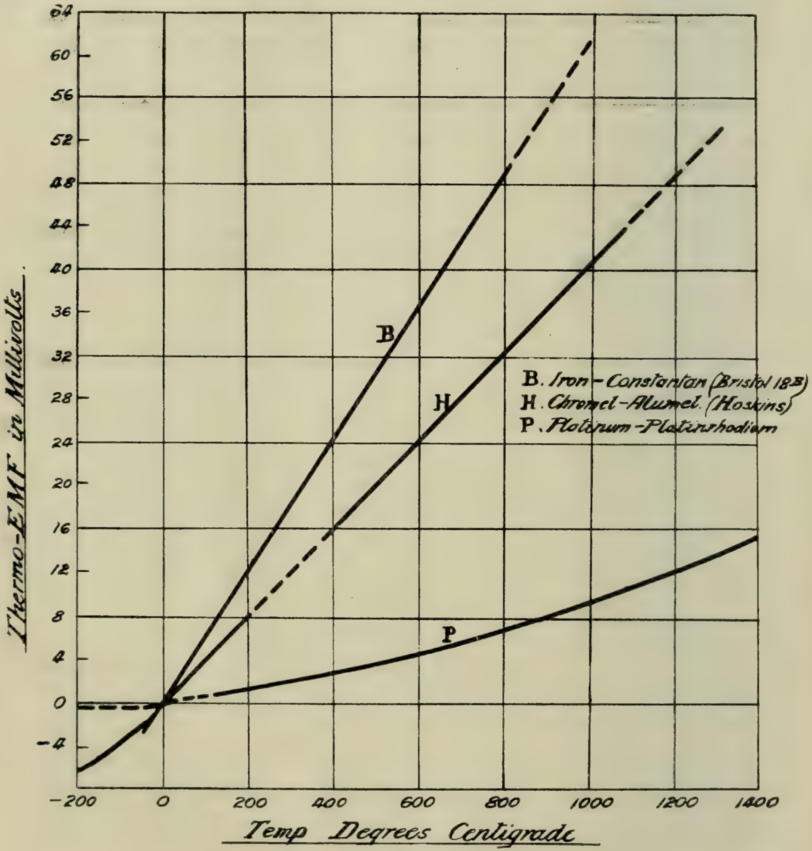

Fig. 3.

"E.M.F.-temp." relation.

(3) *The Linear Relation Between E.M.F. and Temperature.*—This is a desirable property. When the E.M.F. developed per degree rise in temperature is approximately constant over the whole range em-

ployed, the pointer of the dial of the instrument may be set at the cold end temperature, and correct readings are then given over the whole range employed.

Fig. 3 gives the E.M.F. temperature relations of various couples. The graphs shown by some couples exhibit a break due to transformation taking place in one of the metals. Couples of this type are of no use, as there may be more than one temperature which will produce the same E.M.F.

(4) *Permanency and Consistency.*—For reasons of economy it is obviously desirable that the life of couples should be as long as possible, and that they should maintain their accuracy throughout.

It is the usual practice in plant work to place a couple in position and leave it till its period of usefulness is past. The length of this period is entirely dependent on the operating conditions and the type of couple.

Often couples are not checked while in use, and should their calibration alter materially, considerable loss may be incurred through the use of incorrect temperatures. A low-grade product may result, capacity or output reduced, energy or fuel consumption increased, and even destruction of plant may be caused.

(5) *Reproducibility and Ease of Manufacture.*—These are chiefly the concern of the manufacturer. It is essential that couples may be replaced without loss of accuracy and efficiency, and for this purpose absolute homogeneity of the metals is required, as well as simplicity of construction of the couple. Where alloys are used it is necessary that the composition and E.M.F.-temperature relation may be repeated exactly.

Certain types of couple, such as the Hoskins chromel-alumel, have a compensating resistance at the head which serves to adjust any variation which may occur.

(6) *Cost.*—This should not be allowed to assume undue importance, as purchase of cheap apparatus frequently results in inaccurate work and continual renewal.

Each different couple will possess these properties to a greater or less degree, and choice of a couple for any work often becomes a question of sacrifice of one or more to produce the best results for the particular operation in hand. For example, "platinrhodium" couples fulfil conditions (1), (4), and (5) very satisfactorily, but are poor with regard to (2) and to some extent also (3) and (6). Again, an "iron-constantan" couple complies with (2), (3), and (6), but is unable to fulfil (1) completely, as iron oxidizes or forms compounds very rapidly under many conditions at high temperatures.

PRECIOUS METAL COUPLES.

Of this group, the "platinum-platinrhodium" couple is the most important. One wire is of platinum and the other a platinum-rhodium alloy (Pt. 90 %. Rh. 10 %). Platinum iridium is also used, but is not so effective as the platinum-rhodium couples, which are practically indispensable for very high temperature work. The chief advantages of this type are :—

They are absolutely reproducible. Couples manufactured to-day are absolutely identical with those made ten years ago. They resist oxidation perfectly, and this fact, in conjunction with the high melting point of the metals employed, makes them available for use up to temperatures of 1700° C. (3092° F.), provided that they are properly mounted to prevent contamination by foreign matter.

Where great accuracy is required in industrial work, these couples are used because of their reliability, and they are also employed in all high-temperature measurements —1100°–1700° C. (2012° F.–3092° F.)

Their chief disadvantages are :—

The high cost of platinum, which renders it imperative that the wires should be small cross section (usually 0.02 in.) On this account the wires are fragile, of high resistance, and require most careful handling.

The great affinity of platinum for silicon. If these couples are mounted with silica tubes or insulators, they must be protected from all reducing gases at high temperatures. Otherwise there is a tendency towards reduction of the silica and combination of silicon and platinum. Should such combination occur, the calibration of the couple becomes unreliable, and the couple itself becomes brittle and soon cracks.

Effective mountings, relatively impervious to gases and mechanically strong, are essential, as the couples are frail and will stand little abrasion.

The temperature—E.M.F. equation is not linear, and the low E.M.F. developed necessitates the use of a delicate recording instrument.

BASE-METAL COUPLES.

This type finds the most extensive application in all branches of industrial work. The "chromel-alumel" and Brown's "chromel-nickel alloy" probably have the widest field, followed closely by "iron-constantan." Each individual couple, however, possesses its own particular advantages.

In general, the properties common to this class are (2), (3), and (4) given above, and occasionally (1).

Chromel-alumel couples are usually constructed of $\frac{1}{8}$-in. diam. wires insulated either by refractory beads or asbestos string. They have gained repute chiefly because of their resistance to oxidization and slightly higher temperature range. They may be used intermittently up to 1350° C. (2462° F.), but the most satisfactory range for continuous work is not higher than 850°–900° (1562°–1652° F.) Above 650° C. they have a slightly longer life than other types, but after extensive use exfoliate, crack, and become unreliable. Under reducing conditions these couples slowly deteriorate because of the crystallization of one wire. Alumel and chromel wires are somewhat difficult to reproduce, and in the manufacture of the couples a resistance is inserted to compensate for any possible variation in E.M.F.

Iron-constantan Couples.—One wire is of pure iron and the other of constantan, which is an alloy of nickel and copper. The wires are usually $\frac{1}{8}$-in. diam., insulated with asbestos string, and covered with a coat of corundum powder. For temperatures up to 700° C. (1292° F.) this couple has gained much favour, but at higher temperatures the iron wire oxidizes rapidly unless this is prevented by the insertion of some reducing material in the protecting tube.

Iron and constantan wires are more easily produced than other wires, and this is a great advantage, as it eliminates the necessity for a shunt resistance. The majority of Bristol couples of this type seldom exceed 5° (9° F.) from standard.

Furnace work experience of the Electrolytic Zinc Co. at Risdon, Tasmania, has shown these couples to be remarkably durable under certain operating conditions up to 650° C. (1202° F.) They have been found bent and distorted, with inches of insulation stripped, terminals broken or missing, and contact apparently so poor as to be almost unrecognizable as a couple, and yet they were still giving readings within 10° C. (18° F.) of the true value.

Occasionally couples have been found with the hot junctions completely corroded by molten zinc penetrating the protecting sheath. The wires then complete circuit through the zinc, giving rise to a complication of thermo-junctions, but still almost correct temperature readings were given.

In other cases, couples destroyed in similar fashion have had their junctions completed by a piece of Ni-chrome radiator wire, or even copper wire, bound round the ends by an operator. They were then put back into place, and still gave fairly satisfactory readings. Temperatures within 15° C. (27° F.) of the true value have been registered, usually on the low side.

If temperatures are very high this is not always the case, as the wires oxidize, and this causes bad contact, resulting in very low readings.

Above 700° C. iron-constantan couples require an excess of attention and protection if used continuously, though they may be employed up to 800° for intermittent work.

If broken, this type of couple is readily welded without much loss of accuracy.

The Brown Ni-chrome Couple, made by the Brown Instrument Co., has one wire an alloy of nickel (90 %) and chromium (10 %), and the other an alloy of nickel (98 %) and the balance Mn, Al, and Si. This couple has been developed of recent years, and has proved very satisfactory for temperatures up to 1000° C. (1832° F.) It is very resistant to oxidation, and retains its accuracy over long periods.

There are several other types of base-metal couples, but in general they possess a minority of the properties desirable, and are useful only for special purposes.

Couples, as usually manufactured, consist of two wires flame-welded at the hot junction, with each wire insulated with asbestos string or refractory clay bead.

Another type is the so-called " pyod," in which one element is a tube forming a sheath and the other is a wire running down the centre and welded at the bottom.

This type is claimed to possess great sensitiveness to temperature changes, but its disadvantage is that if the sheath is destroyed the whole couple is lost.

Fig. 4 shows the mode of manufacture of several couples. A is a Hoskins chromel-alumel type, twisted at the welded end and insulated with fireclay. B is a Bristol iron-constantan couple, asbestos-insulated. C is the "pyod" type, the outer element of iron serving

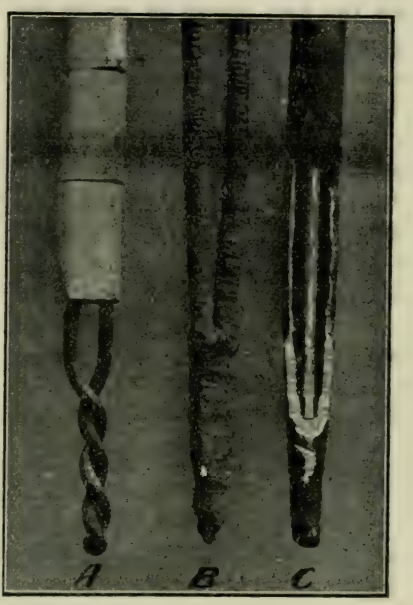

FIG. 4.

as a sheath; the inner element is of constantan, and the two are welded at the point.

COLD END OF COUPLES.

Since the E.M.F. developed by a couple is dependent on the difference of temperature between the hot and cold ends, it will be realized that if the true temperature of the hot end is to be obtained the cold end must be kept at zero (0° C.) In plant operation this is impossible, and here a correction must be introduced by adding to the observed temperature the temperature of the cold end above zero.

If accurate readings are to be obtained, careful attention must be paid to the maintenance of the cold end at a constant low temperature.

The terminals of the couple itself will be frequently close to furnace walls, exposed to hot gases, etc., and under these circumstances it is impossible to maintain the cold end at a sufficiently low figure. This trouble is eliminated by inserting, between the couple and the leads, extension wires of the same metals as the couple. The junction between the couple metals and the copper leads is then at a sufficient distance away from the heat source to ensure the retention of the required low temperature.

This introduction causes a complication of thermo-junctions at the couple head, but these neutralize one another, leaving the whole system equivalent to two unbroken couple wires.

The E.M.F. opposing that at the hot junction is set up at the junction of the couple metals with the copper leads. Actually there are two—constantan-copper and copper-iron—but they are equivalent to a junction iron-constantan.

Many devices have been used for maintaining a constant temperature at the cold end. One is to carry the extension wires back to a hole in the ground some feet deep, where any variation in temperature is extremely slow.

Cold end trouble is most experienced where a number of couples are connected to one instrument, and all have different cold-end temperatures.

PROTECTION OF COUPLES.

The life of every couple is entirely dependent on the extent of its protection. The most important factors in the thermo-electric pyrometry are proper protection for the hot junction and care of the couple itself. Too much emphasis cannot be attached to these points.

Under severe operating conditions protecting tubes are very necessary, to resist heat, oxidation by various gases, fumes, dirt, etc., and the properties required of such tubes may be summarized as follows :—

(1) They must have sufficiently high melting point to resist fusion at the temperature to which they are exposed.
(2) Possess sufficient strength to maintain their own weight at the maximum temperature.
(3) Resist oxidation and corrosive action of molten metals.
(4) Cause no contamination of the couple.
(5) Be relatively impervious to the atmosphere in which they are used. This condition applies chiefly to protectors of platinum-rhodium couples.

Experience at Risdon in furnace and general plant work has afforded the following information on the matter, using chiefly iron-constantan couples.

"Bristol" couples are supplied in an iron sheath, of fairly stout wall and well made. They give good results in furnace gases up to 650° C. (1200° F.), but at higher temperatures the length of life is greatly diminished. A tube may last years when used from 400° to 500° C., but from 650° to 750° C. it lasts only a few months, or even weeks. The portion exposed to the heat soon oxidizes, swells, and finally disintegrates.

When a tube is rendered useless it should be removed and replaced, but this often results in breakage of the couple insulation and consequent oxidation of the iron wire. The life of a couple is much increased if it is left untouched, as frequent changing of tubes invariably causes insulation and oxidation troubles at high temperatures.

In one furnace operating at Risdon much trouble was experienced, and many good couples destroyed. Attempts were made to eliminate the trouble by the use of fused silica tubes closed at one end, but the results were uncertain and uneconomical. Apparently an alteration in crystalline structure of the silica took place, and the tubes became very brittle and had a tendency to crumble. They could only be used with success in a vertical position, and could not stand abrasion.

Experiments were then made with iron tubes filled with various materials. The function of the filling was to exclude air, thus preventing oxidation, and to form a semi-hard mass at the temperature of the furnace, so that when the tube oxidized and scaled off, a hard core was still left, which served as a protection. For this purpose several mixtures of sand, lime, soda ash, graphite, etc., were tried, and eventually one mixture of 1 part soda ash to 9 parts fine cleaned sand by volume was adopted. This mixture is heated to about 750° C. (1382° F.) to drive off water and decompose any organic matter present, and then finely ground. The tubes were filled to a depth of about an inch with this mixture, and the couple inserted and held in place by a small washer. The tubes were then completely filled and tamped down, and finally heated slowly up to the required temperature.

The filling sintered to some extent, and after a few days became hard in the hot zone of the couple. When the tube oxidized this core remained as a protection, and excluded air well. It caused a temperature lag, which was a slight defect, and it also was only serviceable in a vertical position. When this protection failed the couple was usually beyond repair, but its length of life was greatly increased.

For temperatures up to 600° C. the life of the couple was greatly shortened by oxidation, but the sheath lasted a considerable time, when another filling was employed which did not sinter. It consisted of one part graphite to 8 parts of fine clean sand by weight.

When the sheath was nearly destroyed the filling was emptied out, and the couple removed and placed in a new sheath.

The function of the graphite was to keep the atmosphere about the couple reducing, and by this means the life of some couples was considerably increased.

This mixture is unsafe above 650°, because the graphite has a tendency to combine with the iron, making it very brittle.

At temperatures from 700° to over 800° C. iron tubes as sheaths bend very readily if placed horizontally. This is a source of much trouble, as, if a tube is bent, it is difficult to remove a couple without destroying at least the insulation, and very often the junction itself is damaged. The bending is to some extent dependent on the amount of oxidation.

Calorized iron tubes (aluminium coated) do not oxidize very readily, but are not altogether free from bending trouble. If used in a vertical position they are much superior to ordinary iron tubes up to 800° C., but at higher temperatures are not too satisfactory.

There are several other kinds of tube which give excellent protection, but whose initial cost is high. In view of the well-known heat and oxidation-resisting properties of chrome steel, it was decided to test tubes of this material, and they proved very satisfactory, both as regards cost and service. These tubes are of thick wall, and have a slight temperature lag.

The greatest couple protection problem yet presented at Risdon is that of protecting a couple in a molten bath of pure zinc, at a relatively high temperature, in which ordinary iron tubes have a short life, and must therefore be changed frequently to prevent destruction of the couple. This proceeding is both inconvenient and costly.

Silica and porcelain tubes were found to be too fragile, and occasionally broke when placed in the molten zinc. The chief cause of trouble was molten zinc and zinc dross clinging to the side of the tube above the zinc level, which made it difficult to remove the tubes without breaking. Zinc, too, often splashed up into the hole in the furnace wall through which the tube entered, and froze, resulting in certain breakage.

After an extensive trial it was decided to return to iron tubes.

Numerous coatings of fireclay, asbestos, sodium silicate, and other mixtures have been tried on iron tubes, but all had some defects.

The only remaining solution of the problem was to find some metal or alloy which was not attacked by molten zinc.

Various non-ferrous alloys were tested by immersing small pieces of them in pots of molten zinc for many hours, but all were destroyed.

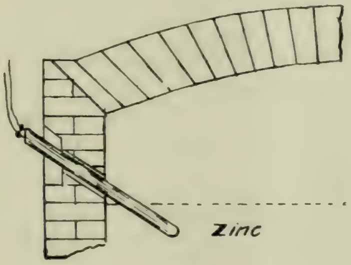

FIG. 5.

The ferro-chromium alloys, although suitable for zinc at relatively low temperatures (430°-450° C.), proved unsatisfactory when used for zinc at relatively high temperatures. A ferro-tungsten alloy gave apparently favourable results for this purpose. However, as ferro-tungsten alloys are exceedingly hard, of high melting point, and unworkable, it was decided to test pure tungsten. A sample of the pure metal was obtained from the manager of the Australian General Electric Co., and proved, after stringent tests, highly resistant to molten zinc. Up to the present, however, it has been impossible to obtain tungsten tubes.

The scheme at present employed with iron tubes is to use two—one of ¾-in. diam., carrying one of ⅜-in., which contains the couple. The outer tube is replaced regularly before it is totally destroyed, and the inner tube protects the couple.

Fig. 5 shows the position of mounting in the melting furnace.

REPAIR OF COUPLES.

In plants where many couples are in use and where facilities for repair work are available, it is economical to collect all burnt-out couples and cut off the damaged portion, which is often merely a few inches at the end. They are then straightened out, re-insulated, and re-welded to form a new hot junction. Old couples which would ordinarily be discarded are thus made use of as many as five times, for, when new, they are generally much longer than is required. An oxy-acetylene flame is quite satisfactory for the welding operation. When preparing couples for welding, the ends should be cleaned, and, if necessary, twisted several times, as in Fig. 4, Couple A.

Repairs of this kind have proved both satisfactory and economical at Risdon, and no couples are now discarded on this plant till they are diminished in length to about 18 in. unless they are oxidized or cracked.

Couples after repair must be re-tested to ensure that the calibration has not altered. The method in use is to place a dozen repaired couples with one or two standards in an electric tube furnace about 3 in. in diameter and slowly raise the temperature. A standard couple is then connected to one indicating instrument, and in turn all the other couples are connected to a second instrument. By this means readings may be obtained over the whole range of temperature for which the couples are to be used. Any that differ substantially from the standard are discarded. Some are found to vary over part of the range, and others over the whole, but the majority are sufficiently accurate for ordinary use. Those which vary over part of the range may be used in some place where the temperatures employed are in the range over which the couple is correct.

Repaired couples are seldom absolutely reproducible owing to slight changes in resistance, but as a rule they are within ± 7° C.

In general there are few furnace operations requiring closer regulation than this. Where closer temperature control is necessary, new couples may be used whenever one is replaced.

When couples are being continually removed from their sheaths the insulation becomes destroyed, and it is necessary to re-insulate.

This is carried out as follows:—Superior quality asbestos string is wound into small loose coils of about 20 yards each. These are soaked in sodium silicate solution of specific gravity 1.25, and then wound cloesly on the couple wires. The latter are then drawn through finely-powdered alundum cement, which adheres to the wet asbestos, forming a thin coat. The whole is then slowly dried The sodium silicate acts as a binder for both asbestos and cement, and the insulation excludes air fairly well.

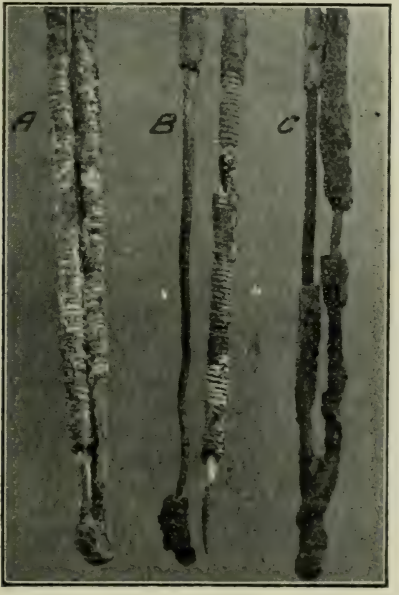

Fig. 6.

It is difficult to insulate an unbroken couple, as the wires are parallel. In this case it is best to bend both wires into the same straight line near the joint. They may then be readily insulated and returned to the original position.

In some cases refractory beads, tubing, etc., are slipped over the wires. These are good electrical insulators, but do not exclude oxygen. This method is quite satisfactory for couples like the "Hoskins" chromel-alumel or the "Brown" chromel-nickel, which are not readily oxidized; but is of no use for iron-constantan over 500° C.

In Fig. 6, A shows an old couple after re-insulation and re-welding. Note the welded junction. B shows the effect of oxidation on the wire, and C broken insulation.

INSTRUMENTS.

As mentioned above, there are two methods of measuring the E.M.F. developed by a couple—the galvanometer and potentiometer methods—the former being most extensively used for technical and the latter for laboratory purposes.

In the galvanometer method a millivoltmeter of the galvanometer type is employed. It consists essentially of a moving coil mounted between the poles of a permanent magnet. Attached to the coil is a pointer which moves over a dial graduated in degrees.

The E.M.F. developed by base-metal couples at their maximum working temperatures is only in the vicinity of 50–60 millivolts, while a precious-metal couple develops only about 10 millivolts at 1000° C.

It is therefore apparent that the instrument must be extremely sensitive; but, on the other hand, it must be sufficiently robust to withstand industrial usage. Improvements of recent years in instrument construction have rendered possible the reconciliation of these conflicting statements, and both requirements are now satisfactorily supplied.

The deflection of the pointer is dependent upon the current flowing through the coil, and this, in turn, upon the E.M.F. developed and the resistance of the circuit. The greater the current flowing the greater the torque on the moving coil, which allows increased strength in manufacture.

By making the resistance of the couple, leads, and the instrument itself low, instruments are possible having a relatively large current flowing through the coil.

The older types of apparatus often consisted of instruments with a resistance as low as 2–5 ohms, and an external resistance—couple plus leads—of 1 ohm. These instruments were very robust, but were subject to errors due to resistance changes, which rendered them unsuitable for precision work, as will be shown below.

If the instrument itself is of low resistance, changes in resistance of the couple leads and connections will cause a relatively large error in the readings indicated, and this suggests the use of an instrument of high resistance. The latter are not so robust, as the current flowing is smaller, and consequently the torque is less, and the coil springs must be delicate.

Experience at Risdon has shown that external resistance troubles are innumerable when a number of couples are used in conjunction with one instrument of low resistance.

Resistance of the couples changes with age and temperature, that of the leads with length and temperature; and further complications are introduced by bad contacts, unsoldered joints, etc.

The relation between the E.M.F. Eo indicated by an instrument and the true E.MP of the couple is given by

$$\text{Eo} = \frac{\text{Rg}}{\text{Rg} + \text{Rl} + \text{Rc}} \times \text{E}$$

Where Rg is the resistance of the instrument.
Rl ,, ,, ,, ,, leads.
Rc ,, ,, ,, ,, couple.
E ,, ,, true E.M.F. of the couple.

Instruments are calibrated so that Eo indicates the true E.M.F. of the couple. Should an alteration take place in the external resistance (Rl + Rc), a corresponding variation occurs in the reading of the instrument.

If Rg is large compared with Rl + Rc, the ratio Rg (Rg + Rc + Rc) is practically unity, and a small variation in Rl + Rc does not materially alter the reading; but if Rg is not large compared with Rl + Rc, a small alteration of the latter may cause a large error in the reading.

This may be better appreciated from an example.

Assuming that the resistance of an instrument, leads, and couple are respectively 5, 1.5, and 0.5 ohms, then

$$\text{Eo} = \frac{5}{5 + 1.5 + 0.5}\text{E} = \frac{5\text{E}}{7} = 0.7143\ \text{E}.$$

If, now, an increase of 0.5 ohm takes place at a contact,

$$\text{Eo} = \frac{5}{5 + 1.5 + 0.5 + 0.5}\text{E} = \frac{5\ \text{E}}{7.5} = 0.6667\ \text{E}.$$

The difference in Eo is therefore .0476 Eo, or 6.63 %. At 1000°; there is then an error of 66°.

Assuming now the instrument resistance to be 400 ohms and the other resistances as before,

$$Eo = \frac{400}{400 + 1.5 + 0.5} E = \frac{400\ E}{402} \text{ or } 0.995\ E.$$

If, then, an increase of the external resistance takes place as before,

$$Eo = \frac{400}{400 + 1.5 + 0.5 + 0.5} = \frac{400\ E}{402.5} \text{ or } 0.994\ E.$$

The percentage error is therefore only 0.1, which represents the negligible error of 1° at 1000°.

The following tabulated figures, given in *Proceedings of American Inst. of Min. and Met.*, 1920, show the comparative errors between two instruments, one of 10 ohms resistance and the other of 300 ohms, each being calibrated to read correctly with a line resistance of 2 ohms.

EFFECT OF CHANGE IN LINE RESISTANCE.

Line Resistance in Ohms.	% Error in Reading.		Error in Deg. C at 1000°.	
	300 Ohms.	10 Ohms.	300 Ohms.	10 Ohms.
1	+0.33	+9.1	+3.3	+91
2	0	0	0	0
3	−0.33	−7.7	−3.3	−77
4	−0.66	−14.3	−6.6	−143

Changes in line resistance are not readily avoided, as they arise chiefly from bad contacts and changes of temperature along the leads. High resistance instruments are practically independent of such errors, but those of low resistance may possess quite unsuspected errors till the apparatus is tested.

An instrument of high resistance and fairly large torque may be made by winding the coil with a large number of turns of fine copper wire, but the temperature resistance coefficient of copper is high, and any temperature change causes a variation in the instrument reading.

In the last ten years great progress has been made in the manufacture of high-resistance instruments of fairly robust type and low temperature resistance coefficient. If the coil is wound with wire of a special aluminium alloy of small cross section (0.003 in. diam.) of low temperature resistance coefficient, the number of ampere turns can be very high, while at the same time the weight is not excessive. This provides high resistance, large torque, and the springs may be made relatively strong.

These instruments are made with resistance ranging from 300 to 1200 ohms, and altogether eliminate resistance troubles. The majority of instruments now on the market are of the high-resistance type.

Instruments used in plant work should be rigidly fastened to a wall or pillar free from vibration, and should also be protected from dust and fumes by a wooden case with glass front.

Automatic recording instruments of the galvanometer type are in extensive use, and are of great value when permanent records are required over any period of time.

METHODS OF STANDARDIZATION OF COUPLES AND INSTRUMENTS.

Base-metal couples need frequent standardization. An error occurring in a couple in use is hard to locate unless it is possible to test it against a standard couple under parallel conditions.

Errors in instrument readings are readily found by comparison with a standard testing instrument.

In practice it is not always certain whether a standard couple placed alongside another in operation is under quite the same conditions. Factors such as distance inside the furnace or length of time in position may make a difference.

As it is rather difficlt to check all couples in use, it has been found advantageous to change them periodically, and standardize those that have been in use, in a tube furnace by the method given above.

The frequency of checking will depend on the accuracy desired. For rough work to within 25° C. it is sufficient to check the couples whenever they are taken out for repair or other purposes.

On a large plant it is desirable to retain one instrument and several standardized couples entirely for plant testing and standardization, and all instruments and leads should be tested at least once a month. Before each round the testing instrument should be checked by immersing a couple connected to it in a bath of metal just above its freezing point and determining the latter by the well-known "cooling curve" method. This provides a fairly convenient and accurate check. The baths used are zinc, antimony, or pure common salt.

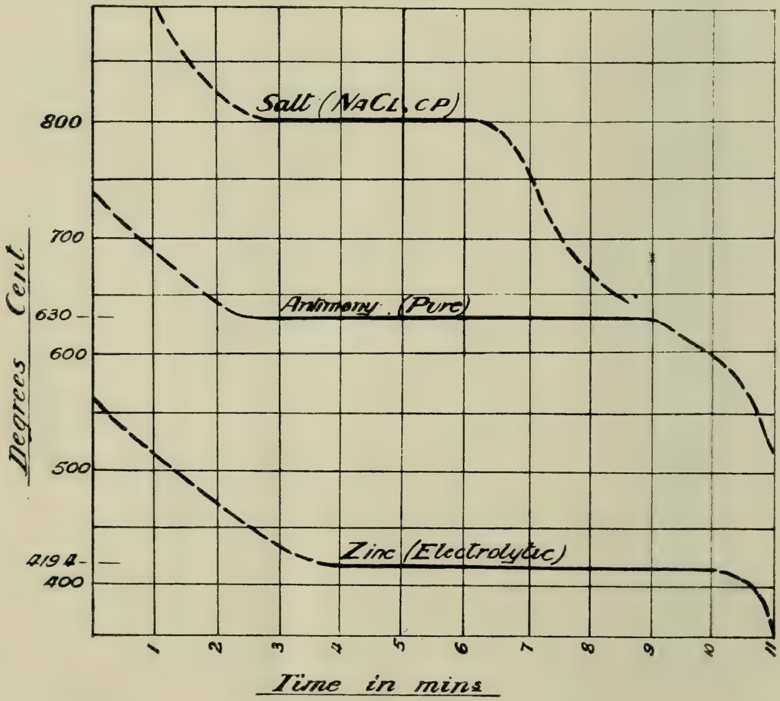

FIG. 7.

TIME-TEMP. COOLING CURVE.

To eliminate possibilities of error during standardization, it is advisable to use two couples in the one tube and interchange them.

Iron tubes may be used as sheaths, but the bath becomes contaminated with iron, and cannot be used many times, as its melting point will alter.

Fig. 7 gives the "cooling curves" of zinc, antimony, and common salt.

Choice of Apparatus.

In any thermo-electric pyrometry installation it is highly essential to give detailed considerations to the type of couple employed. The couple is the one portion of the apparatus causing capital expenditure for upkeep, as replacements are continually necessary, and hence each and every factor concerned should be thoroughly considered.

If the installation is a large one, and conditions varied, it is often advisable to maintain more than one type of couple to meet all existing conditions.

Too much attention is occasionally paid to catalogues, which are all prone to give least consideration to the most important part—the couple. Most catalogues elaborately detail the merits of their respective instruments, but omit everything of importance concerning their couples.

Good instruments are readily obtained after you have chosen the best suited couple.

Cost should not be allowed to influence unduly the choice of instruments, as it should be remembered that good instruments retain their accuracy over years of service if given proper treatment. The same statement applies equally well to leads, switches, and other parts of the apparatus, as the best are always found to give the greatest satisfaction.

Various types of switches are in use, such as plug snap, rotary pole, etc., and all are about equally serviceable. If used with low-resistance instruments it is essential that switches should also be of low resistance and easily cleaned.

Leads should be of large cross section, and well insulated. If a number of couples are connected to one instrument it is advisable to use the one set of leads supplied with the instrument for one couple, as errors are then more readily detected.

In conclusion, the author desires to express his appreciation of the papers on pyrometry which appeared as a symposium in the publication of the American Institute of Mining and Metallurgy, as these dealt in a very full manner with the whole subject of pyrometry. "The Measurement of High Temperatures," by Burgess and Le Chatelier, has also been perused with great interest, and this comprehensive work may be recommended to all serious students of the subject.

Thanks are due to the Electrolytic Zinc. Co. Pty. Ltd., Risdon, in whose research department the work has been carried out, for permission to publish this paper, and grateful reference must also be made to Mr. F. B. Richardson, B.A. (Oxon.), for his kindly assistance in preparing the paper for publication.

THE USES OF CADMIUM.

By H. C. URQUHART, M.Sc., A.I.C.

INTRODUCTION.

ALTHOUGH discovered as long ago as 1817, by Strohmeyer, the use of cadmium did not become in any way general for nearly a century. Its properties, although well known, did not recommend it to users of metals generally, and it was, on the whole, regarded more in the light of a nuisance to zinc producers than anything else.

Its real value was not appreciated until about 1917, when both the United States of America and Germany, who are entirely dependent on outside sources for their supplies of tin, were faced with the unpleasant fact that their tin stocks were being rapidly depleted, with practically no prospect of having them replenished.

This resulted in serious efforts being made in the United States of America to conserve what tin was available, and, as a consequence, several metals were tried in the hope that they would prove satisfactory substitutes for tin. Among these was cadmium, and it was found that cadmium could replace tin in a manner sufficiently satisfactory to permit of its use in many directions where tin had been used before.

It is on this ability of cadmium to replace tin that its use became in any way general at first ; but since then new avenues of development have been opened up in which its physical and chemical properties are proving their value. It is highly probable that its use will extend very considerably as time goes on.

OCCURRENCE OF CADMIUM.

There are several cadmium minerals known, but none of them occur in profitable quantities as ores.

The cadmium of commerce is derived almost entirely from zinc minerals and ores in which it is almost invariably present in proportions which are usually very small. Cadmium behaves metallurgically almost the same as zinc, and hence constitutes a fraction of 1 % of almost all spelter.

The sources of cadmium that have been utilized are zinc ores treated by fractional distillation, lead furnace bag-house "fumes," and residues from the purification vats of electrolytic zinc plants and lithopone plants.

At Risdon, where the ore is obtained originally from Broken Hill, the cadmium is present in the original ore to the extent of 0.03 %.

During the purification of the zinc solutions a product is incidentally obtained containing 10 % cadmium, which is separately worked up for the production of the metal.

PROPERTIES OF CADMIUM.

As the uses to which cadmium can be put in commerce and industry are directly dependent on the properties of the metal itself, it would, perhaps, be well to set out a few of the more important properties which are given herewith.

PHYSICAL PROPERTIES.

The metal has a white colour, and is capable of taking a high polish. It shows no cleavage, and the texture is compact. The fracture is brilliant and crystalline, but this becomes fine-grained and dull when the metal is impure.

The specific gravity of cast metal is 8.546, which is increased to 8.667 by hammering.

It is malleable and ductile, and can readily be hammered into foil or drawn into wire at ordinary temperatures. At 80° C. it becomes so brittle that it can be pulverized.

The hardness of cadmium is greater than that of tin but less than that of zinc.

It has a higher tensile strength than zinc, the figure being stated as 13,660 lb. per sq. in., with an elongation of 44 % in 3 in.

The melting point is 321.7° C., and the metal boils at 778° C., giving a yellowish-brown vapour.

CHEMICAL PROPERTIES.

The atomic weight of cadmium is 112.4. The metal is not attacked by dry air, but in moist air it is gradually covered with a greyish-white film, which is due to superficial oxidation. At red heat it burns to the oxide CdO.

It does not react with water at 100° C., but the vapour is said to decompose steam, with the formation of CdO and hydrogen.

Sulphuric and hydrochloric acids react with cadmium metal only very slowly, as also does acetic acid, but precipitated spongy cadmium is fairly easily dissolved, particularly at the boiling-point of the acid. The usual means adopted for the production of the sulphate or chloride is to react with acid on the oxide, which dissolves readily with the evolution of heat.

Nitric acid dissolves the metal readily. It combines directly with chlorine bromine and iodine in solutions of these elements.

Cadmium is less electropositive than zinc, and is therefore precipitated from its solutions by that metal.

It possesses to a marked degree an ability to form alloys with many of the metals most commonly used, giving, as a general rule, an alloy whose melting-point is considerably below that of the constituent metals. It is upon this property that much of its usefulness in the past has depended, and, in all probability, more of its usefulness in the future will depend.

The metal gives a brilliant and characteristic spectrum, and its use has been suggested as a convenient standard in refractometry.

CADMIUM ALLOYS.

(*a*) *Low Melting-Point Alloys.*—Until comparatively recent times the principal use to which cadmium metal as such was put was the formation of alloys of low melting-point.

These alloys are used for special purposes, the most important being for protection against fire, excessive heat in boilers, and for the production ot special work where the melting-points of the usual metals of commerce are too high.

A series of these low melting-point alloys is given herewith :—

	Bismuth.	Tin.	Lead	Cadmium.	Melting Point.
Lipowitz alloy ..	15	4	8	3	63° C.
Fusible metal ..	4	1	2	1	65° C.
Wood's alloy ..	5	2	4	2	71° C.
Fusible alloy ..	8	3	8	10	75° C.
.. .. ..	16	3	11	2	77° C.
.. .. ..	—	2	1.	1	86° C.
,, ,, ..	3	2	—	1	95° C.

Lipowitz alloy can be bent, hammered, and worked in a lathe. It is used for making casts of small animals and insects, and as a solder for tin and lead, and especially for Britannia metal.

One direction in which the use of Wood's alloy has greatly increased of late is in providing the settings for the crystal detectors used in wireless receiving sets which have become so popular.

Alloys used in the manufacture of fusible plugs in automatic fire extinguishers and safety plugs in boilers have the following composition :—

Lead.		Tin.		Cadmium.
50	..	27.5	..	22.5

A similar alloy is used for the manufacture of cliché metal for stereotype plates. It is better than the corresponding bismuth alloy, as it has a lower melting-point, is harder, and gives a greater number of impressions.

The composition of the alloys used for this purpose varies somewhat, but the following examples may be taken as typical :—

Lead.		Tin.		Cadmium.
50	..	27.5	..	22.5
46.1	..	33.2	..	20.7
35.5	..	52.5	..	12.0

In the manufacture of fuse wire for electric installations the use of cadmium has proved successful. The composition of one alloy which has been used with success is given :—

Cadmium.	Lead.	Tin.	Bismuth.
15	20	21	50

By varying the bismuth content of the alloy the melting-point can be varied to suit special requirements.

BEARING METALS AND ANTI-FRICTION METALS.

When the conservation of tin became a matter of urgent necessity in the United States, attempts were made to substitute other metals or tin in the production of bearing and antifriction metals. Earlierf investigations had already reduced the proportion of tin in bearing bronzes down to very small limits, the proportion of tin being as low as 5 % in some cases, and that of lead as much as 30 %, the remainder being, of course, copper.

In searching for a substitute for the usual copper-lead-tin alloy, it should be borne in mind that a suitable metal should possess the following properties :—

(1) It should be sufficiently rigid to support the load or to resist impact, but yet not so brittle that it will easily crack.

(2) It should have as great a yielding or plastic nature as is consistent with its ability to support the load or resist the impact without the deformation of the bearing as a whole.
(3) The ideal structure combines a hard matrix to support the load and a softer metal or alloy contained within such matrix, to permit the bearing surface to adjust itself to irregularities of service.
(4) It should be easy to handle in foundry and machine shop.
(5) It should be capable of re-melting without deterioration.
(6) For use in babbitt-lined bearings it should be capable of being tinned so that the babbitt can be applied thereto.
(7) It should have good heat conductivity, in order to dissipate the heat generated by friction.

The first metal that was tried as a substitute for tin was antimony. This metal combines readily with copper and lead, and has the property of adding hardness. However, this hardening effect is obtained with the sacrifice of ductility. This leads, in most cases, to fracture under load.

Nickel has been used as a partial substitute with moderate success. Zinc is unsatisfactory, and aluminium worthless.

It is known that a small proportion of cadmium when added to copper increases the tensile strength of the latter, and its alloys with lead do not possess the coarse grain and low ductility displayed by antimony-lead alloys. It increases the hardness of lead to a marked extent, but there is no very great alteration in the ductility of the latter metal.

It has been claimed that cadmium-lead alloys can be used with success for the production of anti-friction bearing metals. These alloys are hard and fine-grained.

A series, tabulated below, has been developed to suit the various conditions under which an anti-friction alloy would have to operate.

	Lead.	Cadmium.
	%	%
For cheap work and light loads	97.25	2.75
For general work	92.00	8.00
For bearings subject to shock	89.00	11.00
For armature pin ends	86.00	14.00
For extremely severe loads .. .	82.00	18.00

It is known that cadmium and lead are not mutually soluble in each other except when the percentage of cadmium is below 5; consequently, all of the above alloys, excepting the first, consist of a mechanical mixture of cadmium with a saturated solution of cadmium in lead.

On acount of the readiness with which cadmium oxidizes at the temperatures at which these alloys melt, precautions are necessary to prevent such oxidation during preparation, and especially during re-melting. When once oxidation commences, it rapidly spreads through the entire melt, thereby rendering it useless, as the oxide is very difficult to remove by fluxing.

The usual fluxes employed are (1) a mixture of potassium cyanide and caustic soda and (2) resin. The function of the KCN is supposed to be due to its ability to reduce the oxide of cadmium as soon as formed.

Resin functions by virtue of its ability to keep the surface of the molten metal bright and clean by removing oxides and dross from the surface as soon as formed.

Copper Alloys.

The alloys of cadmium with copper, although of comparatively recent development, show considerable promise of more extensive application in the future.

The addition of from 0.5 % to 1.2 % of cadmium to pure copper increases the tensile strength of the latter. It gives an increased annealing temperature, greater hardness, and relatively high electrical conductivity (as compared with pure copper wire).

This alloy has been used with success for wireless aerials, long-span transmission wire, and overhead trolley wire. It is claimed that the life of the trolley wires is very considerably increased by the addition of the cadmium.

It is likely that the use of cadmium for this purpose will considerably increase in the near future.

Solders.

The utilization of cadmium as a constituent of solder is another development for which the war was directly responsible. The efforts made in the United States of America to conserve their supplies of tin, and the total inability of Germany to obtain that metal, compelled both countries to seek for substitutes. The use of solder has become so general that it is practically impossible at the present time to do without it. It then became a problem to secure a substitute which yielded alloys resembling those of tin as closely as possible, Investigation proved that cadmium was probably the most satisfactory, and, as it was produced in quantity in both countries most affected by the tin shortage, it is natural that a series of cadmium solders was developed.

This gives a brief outline of the economic conditions prevailing at that time, which far outweighed every other consideration. At the present time, when the world's markets are more or less open to all, and economic conditions are approaching normal, another aspect of this subject demands serious consideration—viz., the technical side.

In the past tin and tin alloys have been used to give certain practical and also certain artistic results. The practical factors involved are those of proper adhesion, continuity of surface, protection, the right degree of hardness, proper working qualities, etc. The artistic results are those that appeal purely to the eye or to our cultivated sense of what is desirable.

Manufacturers of solder have paid great attention to the appearance of their fabricated bars, ingots, or cast sticks in order to make them more saleable. The result is that the user of solder is very apt to judge its quality by the appearance of the rod.

Although this seems to be a comparatively unimportant matter, such is not the case. Tinsmiths and manufacturers of tinware, who are most probably the largest users of solder, have great objections to using any substitute solder that has not the bright appearance to which they are accustomed. Other users do not appear to be quite so particular, but they, too, are disinclined to use anything but that to which they are accustomed.

SUBSTITUTE SOLDERS.

The solders obtained by substituting other metals for tin may be conveniently divided into two classes—

(1) Those in which tin has been entirely eliminated.

(2) Those in which tin has only partly been replaced by another metal or metals.

(1) What is probably the most important example of this class is that having the following approximate composition :—

Lead.		Cadmium.		Zinc.
90.5	..	8.0	..	1.5

It was a well-known fact that, during the war, the German manufacturers had developed a solder which contained cadmium but no tin. It is probable that their product was approximately the same as that given above, which was developed independently by the research department of the Westinghouse Electric Co. as the most satisfactory for their type of work.

(2) When the tin of the ordinary lead-tin solder is only partially replaced by cadmium, very promising results are obtained.

The United States Bureau of Standards, in their Technologic Paper No. 109, published a table setting out the main characteristics of the solders developed by them. This table is reproduced herewith :—

Composition.			Melting Range, ° C.	Sp. gr.	Equiv. vol. to 1 vol. 50/50 solder.	Tensile strength lb. per sq. in.	% Elongation on 2 in.
Sn.	Pb.	Cd.	° °				
50	50	—	210–149	8.81	1.00	5698	20.3
40	60	—	238–149	9.47	1.07	5820	26.0
37.5	62.5	—	245–149	9.54	1.08	5383	28.8
—	90	10	267–249	11.09	1.26	5000	37.5
10	80	10	254–143	10.35	1.17	5727	52.3
5	85	10	257–141	10.67	1.21	—	—
15	75	10	—	10.26	1.61	5880	41.7
50	25	25	149	—	—	—	—

The solder in general use has the composition 40 Sn–60 Pb, and the American authorities have claimed that, for most purposes, a solder of the composition 80 % Pb, 10 % Cd, 10 % Sn, has identical properties.

The use of Cd–Pb solders is said to be complicated by the ease with which the cadmium oxidizes. It is a known fact that this reaction does not readily occur until the temperature has risen some distance above the melting-point of cadmium. When an alloy of lead and cadmium only is being made it is necessary to take precautions to prevent oxidation, as the temperature is sufficiently high to permit the reaction to proceed once given a start.

In the manufacture of solder it will be found that oxidation usually commences at one point; but, once started, it rapidly spreads throughout the entire melt, thereby rendering it useless. If ordinary precautions be adopted, this can be absolutely prevented. The conditions to be observed are—(*a*) the careful regulation of the temperature by means of a pyrometer; (*b*) the use of a flux. Resin is found to be quite satisfactory for this purpose.

It was found that if no flux was used the cadmium rapidly oxidized between 400°-450° C., but when flux was used it was possible to work at 450° C. without any appreciable oxidation.

The addition of zinc in small quantity appears to reduce this tendency, as also does the presence of tin. It is possible that the action of tin in this respect is purely a mechanical one. It is certainly a fact that cadmium will dissolve very much more readily in tin alloy of lead-tin than it will in pure lead, with which it mixes with difficulty. Further, the presence of tin lowers the temperature at which the cadmium goes into solution, and it is thus kept further away from the critical temperature at which oxidation becomes rapid.

When re-melting the Cd-Pb–Sn alloys, however, it is desirable that precautions against overheating be adopted, and that a flux be always used.

As the result of a series of investigations conducted at Risdon, solders have been developed in which a considerable proportion of the tin of ordinary solder has been replaced by cadmium without any appreciable diminution in the applicability or general utility of the solder. The colour and finish of the solders are not quite so bright as that displayed by the lead-tin alloys, but, apart from this, preliminary trials in all departments where solder is usually used have shown great promise and very satisfactory results.

Other Alloys.

There are other and less important uses to which the alloys of cadmium have been put, but none of them find very wide application in the arts or commerce.

Cadmium with gold, silver, and copper, is used for the production of alloys used for the manufacture of jewellery. By variation of the proportion of each metal a wide range of colour is obtained, but as these alloys are usually very brittle, their use is very limited in extent.

Alloys of silver with up to 20 % cadmium are very uniform and homogeneous, and are suitable for trial plates for coinage and silver work.

Cadmium is used with silver in the manufacture of silver work, as it acts as an efficient deoxidizer. It is usually employed in proportions up to 0.5 %, and it imparts malleability, prevents blisters, and improves the colour of the ware.

In the making of bronze telegraph and telephone wires and cables in France and Italy, a fraction of 1 % of cadmium is used as a deoxidizer. It is also used for the same purpose in making nickel alloys, and it is probable it may serve a like purpose in making brass.

Alloys of cadmium have been used to replace steel in the manufacture of hair springs for watches and clocks. Some of them have many of the attributes of steel, but do not rust, and are non-magnetic.

The composition of two alloys used for this purpose is given herewith :—

	%
Platinum	59.25
Copper	16.88
Nickel	18.25
Cadmium	1.25
Cobalt	2.00
Tungsten	1.87

and

	%	%
Aluminium	90.0	96.5
Nickel	0.5	1.0
Antimony	0.5	5.0
Cadmium	2.5	4.0

Cadmium is also used with the cerium earth metals for the production of pyrophoric alloys. The cerium earth metals form an alloy called "Misch metal," which, with 30 %-40 % zinc, cadmium, etc., forms the pyrophoric alloy.

A zinc-tin-cadmium alloy is also used for soldering aluminium and aluminium bronze. The composition is :—

Cadmium	50 %
Zinc	20 %
Tin	30 %

or it may contain cadmium and zinc only, in the proportions 42.9 % and 57.1 % respectively.

The addition of 0.25 % cadmium to zinc is said to increase the breaking strain, but above 0.5 % it has the opposite effect.

A recent development in the utilization of cadmium is the production of the cadmium vapour arc lamp. In this lamp the cadmium is alloyed with a very small fraction of 1 % of gallium, giving a lamp of long life and very free from clouding by adhesion of the cadmium vapour to the walls of the lamp.

This lamp is used as a source of intense monochromatic red light.

USE OF CADMIUM FOR PROTECTIVE COATINGS.

Electroplating with Silver.—Since the year 1895 alloys of cadmium with a little silver have been used instead of nickel for plating metal parts exposed to corrosion. This alloy affords a better protection to the steel, and does not tarnish easily. The deposited film withstands the influence of the atmosphere better than that obtained in electroplating with silver containing 7.5 % of copper. Up to 25 % of cadmium may be used without affecting the colour or appearance of the plate.

Udylite Process.—This process, which is the subject of United States patents, is one in which metals subject to corrosion are protected therefrom by the deposition on them of a thin coating of metallic cadmium.

This coating is then " fixed " by subjecting the coated articles to a suitable heat, which causes the cadmium to alloy with the metal which is to be protected at the surface of contact. The cadmium coating is then polished.

The method by which the cadmium is deposited is by electroplating. The anode is of cadmium, and the article to be coated is made the cathode in the usual way. The bath is a solution of precipitated hydroxide of cadmium in sodium or potassium cyanide, care being taken during preparation that there is no excess of cyanide in the solution. A concentration equivalent to 25 grm. per litre is preferred. The average current density of 7 amp. to 50 amp. per sq. ft. of cathode area can be used. The average time of immersion is stated to be 3 min.

The articles to be coated must, of course, be cleaned in the usual manner before treatment.

After coating, the plated articles are well washed in water and then placed in an oven whose temperature is maintained between the limits 150° C. and 200° C. The inventors state that it has been their practice to keep the plated articles at a temperature of 185° C. for 3 to 4 hours. The time is not important, but excessive oxidation must be avoided.

This heat treatment is intended to fulfil three functions—viz., it alloys the cadmium coating with the metal of the coated article ; it hardens the deposited protective layer ; and, finally, removes excess cadmium by burning it to oxide if the coating is too thick.

This process was first developed as a means of preventing the corrosion of piano wires and springs, where it appears to be quite successful.

In America its use appears to be extending. In 1922 fifty plants were operating this process, and further plants were being built. It has been applied with success to general hardware, locks, automobile parts, etc., and is, in short, a substitute for galvanizing, where a better appearance than that shown by ordinarily galvanized articles is desired.

Schoop Process.—This process was developed as a means of providing a protective coating to metals subject to corrosion. Its use is by no means confined to cadmium, which is a comparatively recent development, but is mainly used with zinc and bronze.

The protective coating is applied in the liquid form by means of a spray. The advantages gained over the older mehods of galvanizing are, of course, that very much larger objects can be treated than could be handled before, and, further, this treatment can be done *in situ.* For example, a large tank could be treated after assembly and erection, and a continuous rust-proof coating applied. The advantages of this method over the older style of treatment of piece metal before assembly are obvious. The coating can be made continuous, and is less liable to damage than is the case where further work has to be done after the protective layer has been applied.

When used for cadmium, excellent decorative as well as protective effects can be obtained by burnishing or otherwise treating the sprayed metal after application.

This method can also be applied to wood or wooden objects. The metal film adheres quite firmly, and a continuous coating is readily produced.

Cadmium Pigments.

The use of cadmium for the production of high-grade pigments is an important one which is extending yearly.

Cadmium sulphide is marketed, as a rule, in two colours—yellow and orange. These two products are identical in composition, the difference in colour being brought about by regulating the conditions under which the pigment is produced.

These pigments are characterized mainly by their purity and brilliance of colour, which is unaffected by the vitiated atmosphere of the cities, where the sulphur in the air darkens the chrome and other yellow pigments in general use. Cadmium sulphide is quite unacted upon by sulphurous gases, hence its popularity where a brilliant and lasting colour is required. This pigment is the most permanent known.

It has long been used by artists, but until recently the high price demanded has prevented its general use. With cheaper and more efficient methods of production the use of these pigments should extend in the near future.

The principal uses to which it is put at the present are enumerated as under :—

(*a*) As a pigment for high-class rubber goods.
(*b*) As a pigment in calico printing.
(*c*) For colouring glass and porcelain, where a range of colours from yellow to reddish-orange is produced.
(*d*) As artists' colours.
(*e*) As a dye with the use of a mordant.
(*f*) For the production of cadmium lithopone.

The following table, taken from the reports published by the United States Government, will serve to give an indication of the importance of cadmium sulphide. From this it would appear that, in the United States at least, the production of the sulphide is probably the most important individual use to which cadmium is put.

	UNITED STATES.						GERMANY.			
	Metallic Cadmium.			Cadmium Sulphide.			Metallic Cadmium.			
Year.	Lb.	Value. Total. $	Value. Average. $	Lb.	Value. Total. $	Value. Average. $	Lb.	Value. Total.	Value. Average. $	Value. Average. M
1906	300	300	1.00	(*a*)			47,368	36.083	0.76	3.19
1907	14,000	14,000	1.025	(*a*)			72,639	60,757	.84	3.53
1908	8,000	6,000	.75	(*a*)			72,741	48,590	.67	2.81
1909	5,300	2,783	.525	(*a*)			81,982	46,777	.57	2.39
1910	4,700	2,585	.55	(*a*)			90,516	39,309	.43	1.81
1911	26,152	17,566	.67	2,392	1,674	0.70	93,861	53,372	.57	2.39
1912	52,508	39,875	.76	8,998	6,400	.71	94,262	63,507	.67	2.81
1913	54,198	41,838	.77	17,302	12,136	.70	81,993	63,641	.78	3.27
1914	91,409	81,205	.89	22,723	20,241	.89	86,362			
1915	91,415	108,443	1.19	10,624	12,057	1.13	82,177			
1916	135,212	210,931	1.56	22,176	27,942	1.26	127,341			
1917	207,408	305,097	1.47	50,169	70,939	1.41	172,284			
1918	127,164	188,203	1.48	51,702	70,315	1.36	235,542			
1919	99,939	121,926	1.22	31,197	37,436	1.20	85,395			
1920	129,283	151,261	1.17	32,133	37,274	1.16	45,971			
1921	68,501	67,131	.98	65,446	71,336	1.09	47,487			
1922	131,590	143,433	1.09	134,774	152,295	1.13				
	1,247,079	1,502,927		449,636	520,045					

(*a*) No production.

CADMIUM LITHOPONE.

Patents have been granted in America for a process of making a cadmium-zinc-barium pigment that is essentially a cadmium-zinc lithopone. A solution of the zinc-cadmium sulphate is treated with a suitable sulphide, such as barium sulphide, and the precipitate is washed, dried, and ground, yielding a pigment whose colour varies from deep yellow to orange, according to treatment adopted.

The colour of cadmium sulphide is very persistent, and, even when diluted with large quantities of a white pigment, its influence is still evident. Consequently, a very wide range of colours can be obtained, and pigments produced which can compete in price with others in common use which have similar shades of colour.

OTHER CADMIUM COMPOUNDS.

The salts of cadmium find many diverse applications, but none of them are very general, and only a small consumption of cadmium is involved.

Cadmium Sulphate.—As an electrolyte in the Weston and other standard cells. It is also used in medicine for affections of the eye.

Cadmium Nitrate.—As a colour for glass or porcelain in the ceramic industry. It gives a reddish-yellow lustre to the articles in which it is used.

Cadmium Iodide and Bromide.—Used in photography for iodizing and bromidizing collodion plates. They are also used in medicine.

Cadmium Tungstate.—As a paint for fluorescent screens in radioscopy.

Cadmium Salicylate.—As an external antiseptic.

Cadmium Borotungstate.—The saturated solution of this salt has a specific gravity of 3.28, and is used in Klein's solution for the determination of the specific gravity of minerals.

Cadmium Acetate.—Is used in dentistry.

Cadmium Chloride.—Is used for testing pyridine bases in calico printing and dye works. It is also used in photography.

THE ELECTROSTATIC PRECIPITATION OF SUSPENDED PARTICLES FROM GASES.

BY H. C. URQUHART, M.SC., AND K. I. MARSHALL.

THE problem of satisfactorily eliminating the fumes and dust which are unavoidably produced in various metallurgical and manufacturing industries is one which has received considerable attention in the past, and has been the subject of much investigation and research.

The solution of this problem was urgently desired for two reasons, the principal one being the fact that a considerable percentage of the total process losses in many industries was represented by the commercial values of the fumes and dust that escaped from the stack. The other reason is that, in some instances, the emission of the matter is regarded as objectionable.

Various methods have been designed with a view to solving the problem thus presented, and the latest development that has been designed is the process of electrostatic precipitation. Continued investigational and developmental work with this process has already demonstrated its applicability to many industries, and new uses to which the process can be put are being opened up. Its use is already widespread, and the economic success that has attended its application in most instances would indicate that its use will become more general in the future.

It has proved its ability to treat gases over wide ranges of temperature, and has successfully handled various types of fumes whose chemical characteristics differ very considerably. In many processes it has now come to be regarded as standard equipment.

HISTORY OF ELECTRICAL PRECIPITATION.

The first recorded observations on the influence of an electrified insulated wire on gases containing solid particles in suspension was made as early as 1824 by Hohlfeld, in Leipzig. He found that, if he electrified a wire hung in a bottle filled with smoke, the smoke cleared rapidly, and a deposit formed on the sides and bottom of the bottle.

About 1850, C. F. Guitard, of London, again called attention to this phenomenon, but apparently the practical significance of these observations was not realized, and the subject appears to have been forgotten.

During the period 1884–6, Sir Oliver Lodge revived the subject, and carried out the first systematic investigation performed thereon. His idea was to use electric discharges for the dissipation of atmospheric fog and the abatement of the smoke nuisance. However, in the year 1885, in collaboration with A. O. Walker and W. M. Hutchins, he applied his ideas and methods to the precipitation of lead fumes at the Dee Bank lead works, in Wales.

He was unsuccessful, largely on account of the deficiencies in the electrical apparatus and material then available. This was the first attempt at the commercial application of the principles of electrostatic precipitation.

During the same period, and quite independently, Dr. Karl Moeller carried out similar experiments, and was granted German patents, but no commercial application was made of his method.

Nothing further seems to have been done until 1906, when Dr. F. G. Cottrell, then Professor of Physical Chemistry at the University of California, and now head of the Nitration Fixation Laboratory of the United States Bureau of Agriculture, had occasion to repeat Lodge's work while studying the removal of acid mists in the contact sulphuric acid process. The results he obtained prompted him to test the commercial possibilities of the process, and so, in 1906, tests were made at the Hercules works of the Du Pont de Nemours Company at Pinole, California. These demonstrated the commercial applicability of the process conclusively.

The first commercial installation was made at the plant of the Selby Smelting and Lead Co., San Francisco Bay, for the collection of sulphuric acid fumes arising from parting kettles. This plant has been in successful operation ever since.

THEORY OF THE ELECTROSTATIC PRECIPITATION PROCESS.

Briefly described, the process of electrostatically precipitating fume and dust consists of passing the dust or fume-laden gas between two suitably-shaped electrodes, which are charged with high potential, unidirectional electricity. One of these electrodes is so shaped to facilitate electrical discharge; the other to minimize it.

By this means the fume or dust is precipitated at a point from which it can be collected without interfering with the continuity of the process. Removal of the dust or fume is done in practice by rapping, shaking, or washing the electrodes, which causes the dust or fume to fall or be washed into suitable receptacles beneath the precipitator.

It is a well-known fact that, when a gas is subjected to an intense electrical field, it assumes the capacity of becoming a conductor of electricity, and a "silent discharge" passes between the electrodes. Under ordinary conditions, air or gas is considered an excellent insulator; but when subjected to an electrical field the gas particles become "ionized." Gas molecules are normally balanced in regard to their electrical properties; but this balance can be upset by several agencies, among which is the action of an electrical field. This means that the gas molecules have an induced electrical charge which enables them to move under the well-known electro-magnetic laws that like repels like and attracts unlike.

Thus, if the "discharge electrode," which, in its simplest form, is a wire fixed axially in a hollow cylinder and insulated therefrom, is charged negatively, the negatively-charged ions will move away from it towards the positively-charged cylinder, and will ultimately discharge thereon.

In practice this type of apparatus is actually employed. A wire is held axially within a smooth pipe and insulated therefrom. It is found that the best results are obtained when the wire, or "discharge electrode," is negatively charged and the cylinder is grounded.

What actually happens in a precipitator of this type may briefly be set out as follows :—

Immediately surrounding the central electrode, which is connected with a source of high-voltage unidirectional current, the electric field is very intense. Here the gas or air molecules are ionized, and this portion of the pipe space is characterized by a faint luminosity visible at night, and usually termed the "corona discharge." By maintaining this portion of the field at the maximum intensity allowable, it is possible to prevent the solid particles from being deposited on the central electrode to any great extent. Should this happen, the regularity of the corona discharge is interfered with, and ultimately a spark-over results. Further, the efficiency of the precipitator is impaired, because deposition of solid particles alters the shape of the central electrode, and this is liable to localize the electrical discharge.

The solid particles in the gas are thus given an opportunity to get by untreated.

Immediately surrounding the corona discharge is a region where the gaseous ions collide with and attach themselves to the solid fume particles, imparting their charges to them and forming " fume ions." These then acquire the same directed mobility as the gaseous ions, and accordingly move across to the collecting electrode. Here they are discharged and collected as solid (or liquid) particles. Meantime, the gas passes on, freed from its solid particles, and unaffected by further treatment.

Constructional Features of Process: Types of Precipitator.

In practice, the precipitator may assume any one of several forms, but may generally be classed in two types—the plate type and the pipe type.

THE PLATE TYPE OF PRECIPITATOR.

In this type of precipitator the collecting electrodes are plates of suitable metal conveniently spaced, and, of course, insulated from the discharge electrodes. Thus, in the original installation at Selby, the precipitator consisted of a lead-lined flue 4 ft. by 4 ft. in section. The collecting electrodes were lead strips 4 in. wide and 4 ft. long, spaced about 4 in. from each other. Between each pair of collecting electrodes was a discharge electrode, consisting of a lead-covered iron pipe.

The " plate " or " box " type of precipitator is usually a rectangular box or section of flue in which the electrodes are placed. Ordinarily the collecting electrodes are of steel, corrugated to give them sufficient stiffness to maintain the desired spacing accurately. The material, of course, varies with the type of gases to be treated, and must be of more resistant material when corrosive fumes are being treated. The collecting electrodes are always electrically grounded.

The spacing is determined by the conditions peculiar to each plant, such as the nature of gas and fume, and is usually from 4 in. to 15 in. The discharge electrodes may be wires or chains suspended from an insulated framework so that they hang midway between the pairs of plates. These wires are kept taut by weights at their lower ends;

but some precaution, such as insulated guys to the wall of the precipitator, must be adopted to prevent swinging of these electrodes, which may be set up by arcing between them and the collecting electrodes.

In all plate precipitators the flow of gas is so regulated that it is parallel to the surface of the plates, and dampers are usually provided in both inlet and outlet, so that the distribution of the gas through the precipitator can be controlled within limits, or the gas flow stopped.

The plate type of precipitator is one of the most economical in first cost, but it is difficult to maintain uniform spacing between the plates and wires, as the plates tend to warp as the temperature changes with varying gas conditions. Obviously, the arc-over voltage is determined by the minimum distance between electrodes at any point in the precipitator, so that any deviation from a uniform spacing will lower the maximum operating voltage, and consequently the efficiency of precipitation.

PIPE TYPE OF PRECIPITATOR.

This type of precipitator is also widely used. In this the collecting electrodes are pipes and tubes, usually of steel, from 6 in. to 18 in. in diam., and from 10 ft. to 20 ft. in length. The material of which the pipe is made varies, of course, with the nature of the gases under treatment. Lead, or even tile or masonry, may be used if the inner surface is covered with a conducting film of moisture, as is the case in the collection of sulphuric acid mist.

The pipes are mounted vertically, a group of pipes forming each precipitator. Wire discharge electrodes are suspended from an insulated framework, so that one hangs along the axis of each tube. As before described, they are guided at their lower ends to prevent swaying.

The discharge electrodes vary with the type of precipitator, and also depend upon the kind of gases being treated. Wire and chains, as well as lead-covered wire and small rods of various sections, are being used in various installations.

INSULATION OF DISCHARGE ELECTRODES.

In all types of precipitators this is somewhat of a problem, particularly when the temperature of the gases is high or when they contain moisture or corrosive substances.

It is the general practice to place the supporting insulators in small chambers outside the precipitator walls and to extend the end members of the supporting framework through the wall to them. This removes the insulator from the path of the main gas stream, but occasionally causes trouble owing to condensation of moisture on it, owing to the fact of its temperature being below that of the gas stream.

When, however, the treater is under suction from stack or fan, air may be sucked through the insulator housing into the treater, thus helping to prevent the deposition of dust and moisture on the insulator.

As a general rule, the porcelain post type of insulator with the usual corrugated surface is employed.

REMOVAL OF COLLECTED DUST.

The usual method of removing the collected dust is by rapping the collecting and discharge electrodes with a series of hammers, which are mechanically or hand operated.

Obviously, it is necessary to shut one unit down entirely while this operation is being performed, otherwise the whole object of the process is defeated, and a portion of the dust will be carried up the stack with the flow of gas.

The dust is collected in hoppers placed at a sufficient distance below the precipitator electrodes to avoid the gas stream disturbing it. From these hoppers it is collected at suitable intervals.

The length of time between cleanings depends on operating conditions, and varies from one to ten hours. It should be remembered that a clean precipitator operates more efficiently than a dirty one, and the period between cleanings should, therefore, be as short as is convenient for operations in general.

INSTALLATION FEATURES.

As a general rule the precipitator is placed sufficiently high above the ground to permit of the installation of hoppers and flues beneath it. The gases may be introduced at either top or bottom of the pipes, but the former is believed to give a more even gas distribution in the case of hot gases.

Provision should be made for by-passing the gases round any unit to permit of cleaning and repairs.

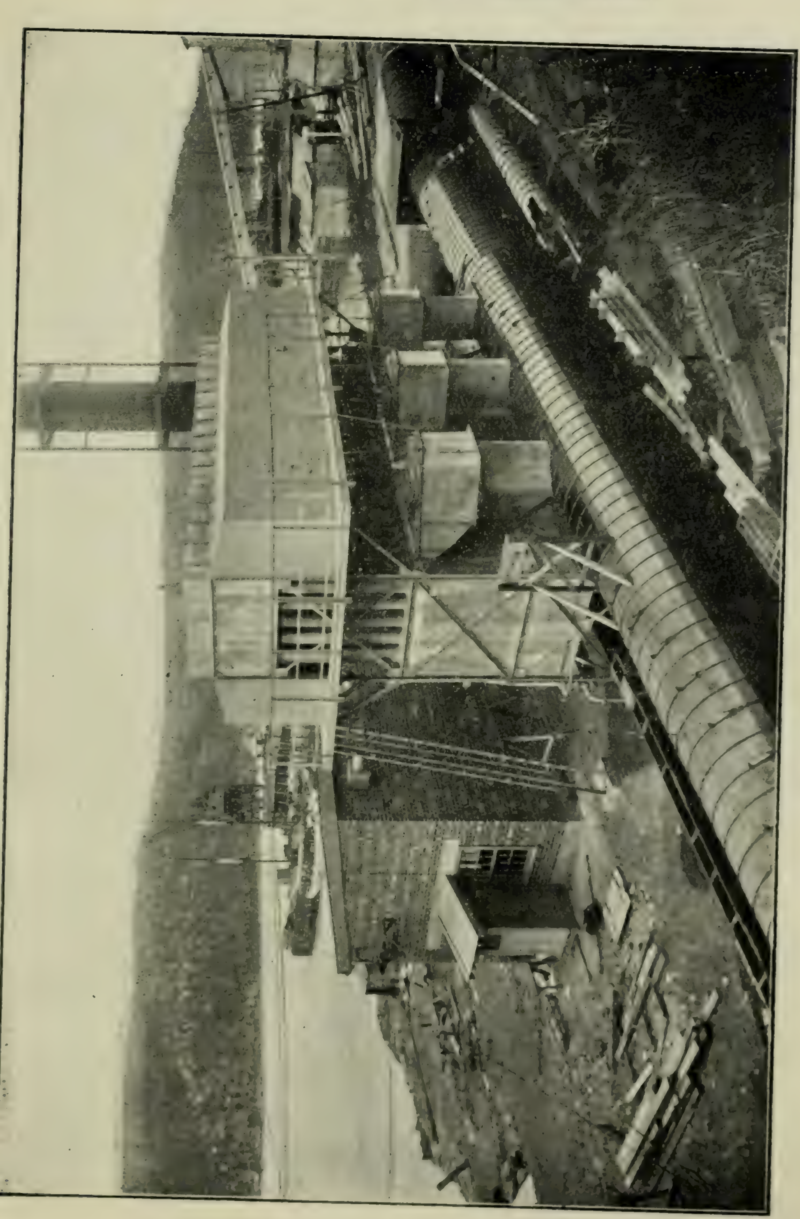

Fig. 1.

Cottrell Precipitator at Risdon Plant of the Electrolytic Zinc Co. of Australasia Ltd.

SIZE OF PRECIPITATOR.

The size of the installation required at any plant will, of course, depend on the conditions under which it will have to operate. The principal governing factor is the volume of the gas to be treated. It is found in practice that in most cases the maximum gas velocity in the precipitator is about 10 ft. per second. Usually it is in the neighbourhood of 7 ft. per second, but this figure often falls as low as 5.

It will readily be understood that high gas velocities tend to decrease the efficiency of the precipitator when it is remembered that, during its passage through the precipitator, every solid particle is being acted upon by two forces. One, the movement of the gas stream, which tends to carry it out of the precipitator, and moves parallel with the electrodes ; the other, the action of the electrostatic field, tends to propel the solid particles across the gas stream in a direction perpendicular to the electrodes. If the former is too great, the solid particle is swept out of the treater before it has time to reach the collecting electrode and be discharged thereon.

For the same reason it is very important that the gas distribution throughout the installation be maintained as uniform as possible.

It is the usual practice to subdivide the installation into units, each one of which is self-contained, and can be cut out without interfering with the continuity of the process as a whole. This has been found to be of great advantage in facilitating cleaning and repairs. Thus, if the whole installation were built as one unit, the buckling of one plate or electrode and the consequent necessity of lowering the voltage would impair the efficiency of the whole plant, whereas, when the installation is subdivided into units, the faulty unit can be cut out, leaving the rest to carry on at full efficiency.

ELECTRICAL FEATURES OF PROCESS.

In all cases a high voltage unidirectional current is used. The reason for this is obvious when it is remembered that the object of the process is to propel all the dust or fume particles to the collecting electrodes. If alternating current were used the particles would merely vibrate to and fro as the direction of the impulse imparted by the electric field changed. This has a tendency to more or less agglomerate the solid particles and thus bring about precipitation, but it is evident that the result is not so good as when unidirectional current is employed.

In actual operation it is found that the best results are obtained when the discharge electrode is negatively charged and the collecting electrode is grounded. This allows a higher voltage to be carried and a smoother operation of the plant.

The voltage employed depends upon the electrical spacing, altitude of the installation, and the character of the gases being treated. It usually falls between 40,000 and 100,000 volts. With high voltages the insulation difficulties increase tremendously, with no commensurate gain.

Power Required.

This depends mainly upon the composition and temperature of the gas under treatment, and so a figure cannot be given that will cover all cases. An average figure is about 2 kw. input for each 10,000 cub. ft. of gas treated per minute, but in some cases the power required falls below this figure.

Very few plants have electrical units of more than 25 k.v.a. capacity each, the great majority being of 10 or 15 k.v.a. The largest in operation are 75 k.v.a. each.

Source of Power.

In all cases the high voltage unidirectional current is obtained by stepping up a single-phase alternating current of low voltage through a special transformer and rectifying the high voltage alternating current produced by means of a suitable mechanical or vacuum tube rectifier.

The low-voltage alternating current is either obtained direct from the plant power circuit or a motor-generator set is installed, usually one for each transformer.

In the latter case, the treater voltage can be varied to conform with the varying gas conditions by changing the generator field excitation. In the former case the high-voltage adjustment is obtained by taps on the transformer low-voltage winding, together with an induction regulator or rheostat, or a combination of both, in the low-voltage circuit, to give close adjustment. The transformer taps are usually so arranged that the high voltage can be varied so as to give 50 %, 62.5 %, 75 %, 87.5 %, and 100 % of the maximum high-voltage rating of the transformer, although this is not rigidly adhered to in all cases.

Where the current derived from the plant power circuit possesses suitable characteristics, a small synchronous motor, deriving its energy directly therefrom, is used to drive the mechanical rectifier. Where a motor generator set is used, the rectifier is coupled to the shaft of the set and driven directly by it. Should the voltage of the plant power circuit vary greatly, or if only direct current is obtained, it would be necessary to instal the motor generator set.

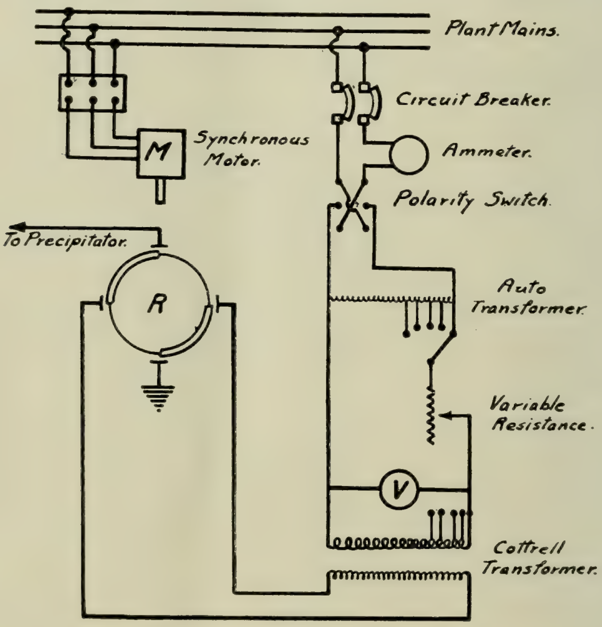

Fig. 2.
Wiring diagram—Synchronous motor system.

One horse-power or less is all that is required to drive the mechanical rectifier.

Transformers.

As the transformer is a most important part of the equipment, it is evident that its construction and protection call for the very highest type of workmanship. When operating in this service, the electric strains are probably more severe and continuous than in any other kind of work. It is probably due in the greater part to the great advances made in recent years in the production of this type of apparatus that the success of the electrostatic precipitation process is due.

The insulation in the coils of the high-voltage windings must be absolutely of the best quality, and the most careful workmanship is necessary in the manufacture of this part of the transformer. Special care must be given to the insulation of the end coils, and particularly to the end turns of the end coils. The high-voltage coils must also be well braced, as the strains are particularly severe.

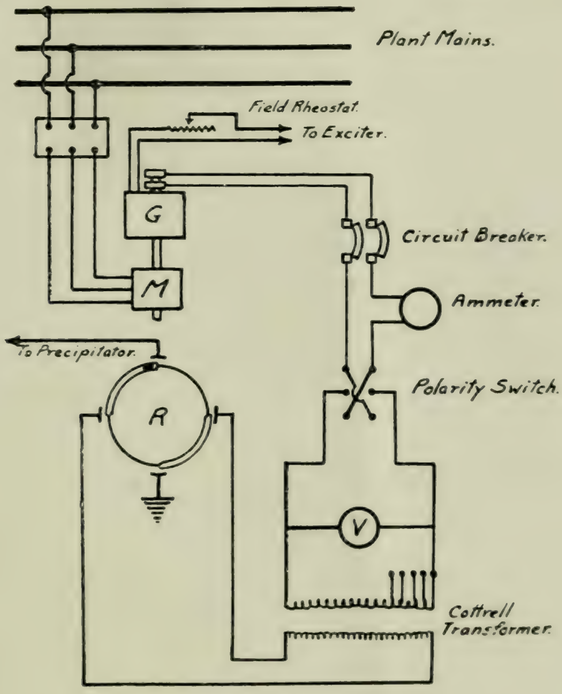

FIG. 3.

WIRING DIAGRAM—MOTOR GENERATOR SYSTEM.

Special precautions are taken to protect the transformer from electrical surges in the high-voltage circuit. As a rule, groups of protective resistances are employed, although the older method of mounting choke or reactance coils on the transformer terminals is still used.

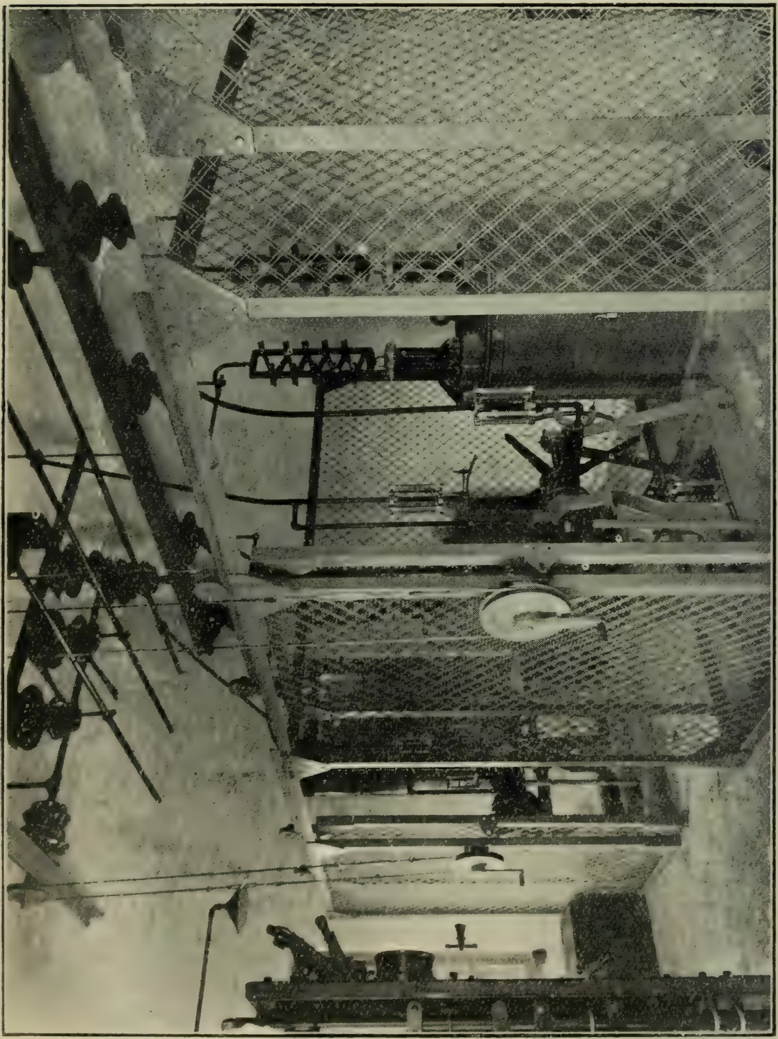

Fig. 4.

Interior of Cottrell sub-station at Risdon plant of the Electrolytic Zinc Co. of Australasia Ltd.

MECHANICAL RECTIFIERS.

In the usual type of rectifier the rotating element has four arms or a disc constructed of non-conducting material. In the case of the arm type, each arm carries a metallic point, and the two opposite pairs of points are connected by a wire. The stationary member consists of four metallic shoes supported on non-conducting material, and the whole structure is so arranged that the position of the shoes can be altered so that any portion of the alternating wave can be rectified or picked off.

In the case of the disc type, the rotator consists of a disc of bakelite or similar non-conducting material with two metallic strips, each 90 mechanical degrees in length, mounted opposite each other on the circumference (see Figs. 2 and 3).

The rotating points or shoes never make actual contact with the stationary shoes owing to their high peripheral speed, which approximates two miles per min., so the high voltage current is forced to spark across the intervening gap.

SWITCHBOARD.

This equipment includes the necessary circuit-breakers, switches, and instruments, including a double-pole double-throw reversing switch in the low-voltage circuit of the transformer. By means of this switch the direction of flow of the high-voltage current through the treater can be controlled, and the negative polarity can be maintained on the discharge electrodes.

PROTECTIVE RESISTANCES.

It is generally advisable to place similar resistance units to those used on the transformer circuit in the high-tension lines between the rectifier and the treater. Smoother operation is thus obtained, and the current surge in case of arc-over is limited by their use.

APPLICATION OF PROCESS AND RESULTS.

The installation of this process has been shown in many instances to prevent losses which have occurred, particularly in such operations as smelting. Here the dust and fume often contain material of considerable value. The following are striking examples :—

Type of service.	Capital cost.	Gas volume c.f.m.	Amount of material recovered lb. in 24 hours.	Value of material recovered per 24 hours.	Operating cost per day.	Remarks.
Bismuth smelter (blast furnace)	£5,000	5,000 c.f.m.	590 lb. 57 % Bi.	£149	£3	Before the installation of the Cottrell, approximately half of these values was recovered in dust chambers and a water scrubber. 90 % effy. Cottrell.
Copper smelter (multiple hearth roasters)	£34,000	125,000 c.f.m.	59,500 lb. 5.4 % Cu. 7.5 oz. Ag.	£69	£6	95 % effy.
Copper smelter (rotary dryers)	£25,000	45,000 c.f.m.	10,000 lb. 26 % Cu.	£58	£5	99 % effy.
Lead smelter, D.L. sintering machines (blast furnaces)	£20,000	103,000 c.f.m.	38,000 lb. 75 % Pb.	£316	£14	97 % effy.

NOTE.—In calculating the above table copper has been valued at 10 cents per lb., silver at 50 cents per oz., bismuth at 2 dollars per lb., and lead at 5 cents per lb. This leaves a considerable margin for the cost of handling the material caught in the Cottrell precipitators.

Power has been estimated at approximately £20 per h.p. year.

P. E. Landolt, writing in *Chemical and Metallurgical Engineering*, vol. xxv. (1921), gives some interesting figures. He states that, at that time, the total investment in electrostatic precipitation plants at copper smelters was 5,000,000 dollars (£1,110,000), over 8,000,000 cub. ft. of gas being treated per minute, with a recovery of 5,000,000 lb. of copper per annum, yielding a net return of 20 % on the investment.

In lead smelters the capital invested in the process was 3,000,000 dollars (£667,000), and the lead recovered totals from 15,000,000 lb. to 20,000,000 lb. per annum.

In tin smelting plants the capital investment in Cottrell treatment plants was 400,000 dollars (£88,900), saving at least 2,000,000 lb. of tin yearly.

For the silver refineries, while the investment in precipitators is not over 300,000 dollars (£66,700), the total recovery of silver alone is 500,000 oz. per annum, in addition to which considerable gold is saved.

It is also proved to be the most satisfactory method of collecting fogs, such as sulphuric acid mist, from acid concentrators. In United States it is installed as an integral part of a standard make of acid sludge concentrator. The object of these installations is twofold—first, the recovery of acid, and secondly, the prevention of the mist nuisance in the neighbourhood of the plant. Several precipitators were installed in England during the war on Gaillard concentrators.

In many cases the fact that the process may be successfully used to clean gases at high temperatures has led to its installation. Notable among these are installations in cement mills for collecting the dust which formerly escaped from the rotary kiln stacks, the cleaning of hot roaster gases used in the manufacture of sulphuric acid, and the cleaning of hot blast-furnace gases in iron and steel plants.

Cottrell electrical precipitation plants were first applied to the rotary kiln stacks of cement plants to prevent the escape of dust into the atmosphere. It is interesting to note that in United States alone approximately 360,000 tons of cement kiln dust were collected by this method in the year 1921.* Later it was discovered that when the raw mix contained feldspar this collected dust was valuable as a fertilizer because of its potash-lime content. In the case of a

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* Evald Anderson—"Cement Mill and Quarry." January 5, 1922.

California cement plant the kiln gases are passed through a spray chamber, followed by a settling chamber, and then through a Cottrell. Practically the only material entering the precipitator is fine particles of the volatilized alkali salts. This method leads to the recovery of a product analyzing from 30 % to 40 % K_2O.

In continental Europe the greatest field for electrical precipitators has been that of cleaning hot roaster gases for use in the manufacture of sulphuric acid. Not less than 16 plants are in operation in Germany and 2 in France.

Fractional precipitation has been used successfully in both mercury and arsenic recovery plants. Above 250° C. the arsenic content in gases is in the form of a true gas. Hence a Cottrell at this point in the flue system may be used to collect any dust particles or impurities. If the gas is subsequently cooled, arsenic is condensed as a fume, and can then be removed by passing through a second precipitator. Much the same process is being applied to gases carrying mercury vapour in a California mercury smelter.

Other uses to which the process has been successfully applied include the tar and oil separation from by-product operations, dust arising from grinding operations, air cleaning in ventilation work, the precipitation of phosphoric oxide for the preparation of phosphoric acid by smelting phosphate rock, and the collection of various finely-divided food products which are dried in spray-type driers.

The United States Bureau of Mines * has performed some extensive investigations on the chloride volatilization process of ore treatment in which the Cottrell process is used exclusively for the collection of the volatilized products.

It is apparent from the foregoing that the application of electrostatic precipitation has been very broad, and that many varying conditions have had to be met by the precipitation engineers. In fact, each individual installation has required careful consideration. Several factors have an important bearing on the size, type, and location of the precipitators, as well as the structural materials to be used in their construction. The type of electrical equipment varies, too, with plant conditions.

These factors are briefly enumerated as follows :—

The average and maximum volume of gas to be treated.

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* Bulletin 1211, Bureau of Mines 923, U.S. Government.

The temperature and composition of the gas.
The barometric pressure at the location of the installation.
The origin of the gases and plant lay-out, giving sizes, type, and arrangement of furnaces of apparatus used, flues, and stacks.
The nature and quantity of fume or dust; and, finally,
The characteristics of the electrical power available at the plant, or, in the case of no electric power being used, what source of mechanical power is available.

EFFICIENCY OF PRECIPITATION.

Where the guiding factor in the design of a precipitator is the recovery of valuable materials, it is possible to calculate the optimum size of a precipitator. On the one hand is the value of material to be recovered at various efficiencies, and on the other hand are the operating costs of power, labour, and supplies, and the interest and depreciation on the capital cost of the installation.*

Where the problem is the prevention of visible fume or dust, efficiency of collection is not as important as the maintaining of a clear effluent gas from the stack.

THE ELECTROSTATIC PRECIPITATOR AT RISDON.

A treatment plant has recently been erected at the Risdon works of the Electrolytic Zinc Co. in connection with the roasting operations.

A pipe type of precipitator is used, consisting of 150 pipes 8 in. in diam. and 12 ft. long, built in three units of 50 pipes each. The whole structure is carried on a steel framework, the plates, pipes, and other parts coming in contact with the gas being made of lead.

The discharge electrodes are made of special lead-covered wire, the covering of which is star-shaped when viewed in section.

Each section is provided with its own unit of electrical equipment, consisting of transformer, arm type of mechanical rectifier driven by a synchronous motor, switchboard arrangements, and protective devices. Fig. 1 gives a fair idea of this equipment.

Each electrical unit is of 15 k.v.a. capacity.

In this installation approximately 33,000 cub. ft. of gas are treated per minute, the gas velocity through the precipitator pipes being approximately 10.5 ft. per second.

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* W. A. Schmidt—Paper presented at the Pacific Coast Convention of the A.I.C.E., Vancouver, B.C., August 8-11, 1922.

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A FEW NOTES ON THE HYDRO-ELECTRIC POWER OF TASMANIA.

By C. C. HALKYARD, M.C.E.*

SYNOPSIS.

A BRIEF account of the water-power potentialities of Tasmania is given, together with possible applications of the power commercially.

A short history of the Waddamana development is then given, and the main works are described.

WATER POWER POTENTIALITIES.

Tasmania is an island of, approximately, 24,500 sq. miles area (exclusive of small islands).

An area of 12,000 sq. miles is at an elevation of over 1000 ft. above sea-level, and of 2300 sq. miles at over 3000 ft. above sea-level.

Average annual rainfall varies from over 140 in. on West Coast to 20 in. on East Coast, and for the whole island averages 48.5 in.

The total water-power available, based on conservative figures for run off and efficiency, is 6,000,000 b.h.p. Only a small percentage of this total will, under present conditions, and under likely conditions in the immediate future, be capable of being developed commercially.

An extensive hydro-metric and water-power survey is at present being carried out in order to collect information concerning the most promising water-power schemes.

Already 400,000 b.h.p. has been located in different proposals, and there is still a large area of very likely country along the West Coast which has not been investigated yet.

Tasmania thus has very vast power potentialities, which, combined with good ports and communications and wonderful mineral and timber resources, should augur well for the establishment of the great key industries, particularly chemical and metallurgical.

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INDUSTRIES.

Already several important industries are located in the State, the foremost being the Electrolytic Zinc Company of Australasia Limited, which is engaged in the extraction of zinc from imported calcines. This company is now taking 30,000 h.p. of electrical energy per annum from the Department. It has also commenced a large new works for the manufacture of superphosphates, and is expected to produce approximately 50,000 tons per annum, involving the importation of 25,000 tons of phosphate rock from Nauru. The company has also extended its operations to the West Coast, and will be a large user of electric power for the treatment of ores in that district.

The Carbide and Electro Products Limited, operating at Electrona, produces carbide of calcium with entire success, and has made electrodes for its own use.

The operating of electric furnaces and electrolytic cells has reached the stage of commercial efficiency, and a wide field is open for their use. The following list gives briefly some of the more important products of the electric furnace and electrolytic cell :—

RAW MATERIALS.	PRODUCTS.	APPLICATIONS.
Anthracite coal ..	Graphite	Electrodes, lubricants, and paints.
Sand, sawdust, and coke	Silicon carbide ..	Abrasives and refractories.
Bauxite (natural aluminium oxide)	Artificial emery ..	Abrasives and refractories.
Bauxite	Aluminium metal	Cable, aeroplane, and auto. alloys, kitchen utensils, alumino thermic reactions, de-oxidizing steel, camp utensils, ammonal (explosive), speaking tubes on naval vessels, acid containers.
Sand and coke ..	Silicon	Silicon steel, hydrogen for balloons, resistance units, silicodes, silicon tetrachloride.
Quartz rock ..	Fused quartz ..	Silica tubes, heat-resisting materials.
Coke and sulphur ..	Carbon bisulphide	Solvent, insecticide, carbon tetrachloride viscos.
Phosphate rock, coke and sand ..	Phosphorus ..	Matches, phosphorous compounds, phosphor-bronze, smoke screens.
Lime and coke ..	Calcium carbide ..	Acetylene for welding, cutting, and lighting ; aceton, acetic acid, aeroplane dope.

Calcium carbide (nitrogen of the air)	Calcium cyanamide	Fertilizer, ammonia, cyanides.
Tin dross	Refined tin	Tin-plate industry, bronzes.
Iron ore	Pig iron	Steel industry.
Iron, silica rock, coke	Ferro-silicon	Making steel, making hydrogen.
Manganese ore and coke	Ferro-manganese	Steel permanganates.
Chrome ore	Ferro-chrome	Special and high-speed steels, armour plate, projectiles.
Tungsten ore	Ferro-tungsten	Special and high-speed steels.
Molybdenum ore	Ferro-molybdenum	Special steels.
Titanium ore	Ferro-titanium	Scavenger in steel-making.
Zinc ore	Pure zinc metal	Brass industry.
Water	Hydrogen and oxygen	Balloons, oxy-welding, oxy-cutting.
Water, salt	Caustic ($NaOH$)	Soap, paper industry, making explosives.
Water, salt	Chlorine gas	Bleaching, explosives, chlor-benzol, gas warfare, mustard gas, phosgene, chlorpicrine, silicon tetrachloride, water purification, sanitation, surgery (Dakin solution), detinning, artificial plastics (halowax), hydrochloric acid, aluminium, chloride for oil-refining.
Water, salt	Hypochlorite	Disinfectants.
Water, salt	Sodium metal	Peroxides, cyanides, bleaching, mining.
Water, salt	Hydrogen	Ballooning, hydrogenate fat (for cooking).
Magnesium chloride or magnesia	Magnesium metal	Flashlight powders, light, Wright alloys, tracer bullets, and flares.
Air	Nitric acid	Explosives, fertilizers.
Air	Ozone	Sterilization of water.
Crude copper	Pure copper	Electrical industry.
Pig iron	Pure or "Swedish iron"	Tubes and special steels.
Potassium chloride	Potassium chlorate	Primers, matches, dyeing.
Chromium	Sodium bichromate	Dyeing.

In view of the great water-power potentialities in Tasmania, it is probable that the chemical and metallurgical industries of the State will greatly expand in the near future. The textile industry is also represented by two large woollen factories in the north of the State, and a further mill may shortly be erected in the Midlands.

Efforts are being made to establish cotton-spinning mills in localities which are specially suitable for this industry. Interests concerned in the paper industry are actively considering the problem of utilizing Tasmanian timbers for the manufacture of wood pulp for paper-making. Experiments have been carried out by the Commonwealth Bureau of Science and Industry which prove that the timber of this State is eminently suitable for the purpose. This industry, when established, will require a large quantity of power. Tasmania is specially rich in timber resources, which, combined with plentiful cheap power, should, in the near future, make Tasmania possibly the largest paper producer in the southern hemisphere.

HISTORY OF WADDAMANA

The development of the Waddamana scheme was the direct result of the demand for cheap power for metallurgical processes. The pioneers in the field were the Complex Ores Company, of Melbourne, under the leadership of J. H. Gillies. This company approached the Government of Tasmania in 1908 with a request for cheap electric power in order to establish a process of zinc extraction from the complex ores of the West Coast of Tasmania. The government decided that the condition of the State's finances prevented any action being taken towards State development of power. A concession was, however, granted to the company to develop power from the waters of the Great Lake, and construction was started in 1911 by the Hydro-Electric Power and Metallurgical Company, formed for the purpose. This company carried on under difficulties until 1914, when the Government acquired the works then constructed for the purpose of developing electric power.

The first stage of the scheme was in operation in 1916, and the extensions to full capacity were shortly afterwards commenced in order to supply a large block of power to the Electrolytic Zinc Co. for the extraction of zinc. The scheme was completed to full capacity towards the end of 1922.

DESCRIPTION OF WADDAMANA DEVELOPMENT.

The main catchment area is 153 sq. miles in area, and drains into the Great Lake, which stands at 3368 ft. above sea-level at full supply level. A subsidiary catchment, that of the Ouse River, is 107 sq. miles in area, and its drainage has been diverted into the Great Lake by means of a diversion canal.

Ouse Diversion Canal.

This canal is approximately 7 miles long, and has a normal capacity of 450 cub. ft. per second. The Ouse River is diverted into this canal in a narrow rocky gorge by a small concrete weir. The first half-mile of the canal is in the form of a reinforced concrete flume 9 ft. wide by 6 ft. high, laid on a grade of 1 in 330. This flume discharges at the rate of 10 ft. per second, approximately, through a transition into the canal proper, which is 24 ft. wide on the bottom, 5 ft. deep, and has $1\frac{1}{4}$ to 1 side slopes.

This canal was cut through clay and boulders and through solid diabase on a grade of 4 ft. per mile. A large amount of rock excavation was necessary, and for this purpose portable compressors, driven by oil and steam engines, were installed. Jackhamers were used for holes up to 6 ft. depth, and tripod drills for deeper holes. Earth excavation was carried out by barrows, scoops, and revolving steam shovels.

Spillways are provided at the headgate and at intervals along the length of the canal to discharge surplus waters.

Great Lake Dam.

Conservation and regulation of the water supply is effected by storing water in the Great Lake. The first stage of the development called for a small weir only, and this was constructed in diabase masonry across the Shannon River at its outlet from the Great Lake.

The extension of the scheme to full capacity necessitated raising the water-level by 35 ft., and for this purpose a dam 40 ft. high was constructed.

A huge reservoir has thus been created of capacity 312,500,000,000 imperial gal. and of area 58 sq. miles at full supply level.

The dam is of the multiple arch type, and has been constructed in reinforced concrete.

Total length is 1180 ft., made up of 27 arches of 40-ft. span and 100 ft. of abutments.

The arches are circular in plan, with constant radius for extrados of 21.947 ft., and thickness increasing from 1 ft. at top by a constant rate of .019 ft. per ft. of vertical depth.

Extrados is sloped 60° with horizontal, and weight of water supported thus adds considerably to the stability of the structure.

Buttresses are on 40-ft. centres, and are tapered. Top is 5 ft. long by 1 ft. 10 in. wide ; sides are on 1 in 40 batter, and downstream end on 3½ in. per ft. batter. A footway was constructed along the top of the buttresses, and two rows of tie-beams were constructed between buttresses in order to stiffen same.

Four sluice-gates, protected by screens, were installed in the central span, and are of sufficient capacity to deal with flood discharges without the necessity of a spillway.

The dam contains nearly 10,000 cub. yds. of concrete, and is the longest of this type yet constructed. It is also the first dam of this type to be constructed in the southern hemisphere.

Constructional difficulties were great owing to long communications of poor character, foundation troubles, and scarcity of skilled labour. The foundation troubles were not indicated by the trial pits sunk previous to construction, and developed only after a considerable amount of work had been done.

The trial pits bottomed on fissured diabase. Previous experience with the foundations of the old dam and with a few of the pits sunk through the fissured diabase to solid rock indicated that the solid rock existed about 2 to 3 ft. below the fissured rock. This assumption was found to be erroneous in the case of two strips—one on either side of the centre, where deep excavation (52 ft. max.) was necessary owing to the fissured rock persisting to a great depth.

The rock was intersected by greasy seams of decomposed zeolite dipping steeply. This made excavation very hazardous, particularly as there was considerable hydrostatic pressure, due to the close proximity of the lake. Heavy timbering had to be used, and, even with heavy timbers closely spaced, signs of failure became noticeable. Overburden had to be removed to prevent caving in of trenches, and it was not possible to further reinforce the timbering without interfering too much with the working space.

A further difficulty presented itself in these deep holes. Ordinarily the arch sat on a circular drum forming a cut-off, and founded on solid rock ; but where the depth was great this method was abandoned, and the arch carried right down to solid rock. The upstream face of the excavation was vertical, and the downstream stepped. Timbering consisted of a series of long struts arranged radially like the spokes of a wheel, and this method proved to be very effective.

Concreting in these deep trenches was costly and laborious, owing to the difficulties in removing timber. Concrete was placed by means of steel chutes from a travelling tower 80 ft. high, which was hauled along a track parallel to the dam by winches driven by air. Stone, sand. and cement were delivered to the tower by trucks and elevated to bins holding one day's supply. Measuring hoppers were located directly under the bins and fed direct to the mixer, which was electrically driven.

Electrical power was delivered from Waddamana at 6600 volts and transformed to 415 volts. Practically all the consruction plant was driven electrically.

Waddamana Canal.

The water, after leaving the Great Lake, flows down the Shannon River for 5 miles, and is then diverted by a small rock-fill weir into the Waddamana Canal.

The intake, which regulates the supply of water to the canal, is a masonry structure with six hand-operated wooden sluice-gates.

The canal is of 600 cusec. capacity, and is 3½ miles long. The upper section is 33 ft. wide on bottom, 1 to 1 side slopes, carries water 5.5 ft. deep, and is on grade of 4 ft. per mile. The lower section is 60 ft. wide on bottom, 1 to 1 slopes. carries water 4.5 ft. deep, and is on grade of 3 ft. per mile. As the natural fall of the country is too steep for these grades, a series of concrete and masonry drops have been constructed in order to check excessive velocities.

The canal was originally constructed for 280 cusec. capacity, and was widened and deepened without interruption to supply. At two deep rock-cuts diversions were made instead of widening, as there was a danger of the canal becoming blocked at these points.

The canal empties into a natural lagoon, the level of which has been raised by long earthen banks. This lagoon acts as a balancing reservoir, and is capable of running the station for several hours at full capacity.

Pipe Lines.

The pipe lines are fed from the lagoon by means of duplicate concrete regulators and screening structures known as the "penstock chambers," or "forebays." Coarse outer screens, consisting of double rows of pickets driven into the ground, with 4-in. openings, serve to deflect ice and timber, while weeds, etc., are dealt with by sliding screens of structural steel and rods. Regulating gates are constructed of hardwood, and are hand-operated.

Water is discharged from the penstock chambers through rivetted steel bell mouths and throttle valves into four wood pipes, one of which is 4 ft. diam., and the others 5 ft. diam. Two of the pipes are constructed of oregon and two of West Australian karri. The pipes are built of staves shaped to the diameter of the pipe, tongued and grooved on the sides, and, for the more recent pipes, on the ends also.

The bands are constructed of round mild-steel bars with rivet heads on one end and about 5 in. of rolled thread on the other. The bands are bent to correct shape of pipe, and the ends are connected by means of a shoe or clip. Spacing and thickness of bands are determined by the head—a factor of safety of 4 being allowed usually.

Numerous fittings, as air pipes, air valves, scour pipes, pressure relief valves, etc., are attached to the pipes where necessary, the attachment being made by means of cast-iron saddles seated on rubber insertion rings and held by bands.

The grade of the upper portion of pipe line is relatively flat, and, in order to provide good regulation and to prevent collapse of the pipe, it was necessary to erect a steel stand-pipe 15 ft. diam. and 68 ft. high at about one-third of the distance along the pipes. The wood pipes continue for a distance of about a mile down to 150-ft. head, and thereafter there is a great lack of uniformity in the style and size of pipe erected. This state of affairs was caused by the impossibility of obtaining the pipes one wanted while the war was on and the necessary acceptance of the only alternatives offering.

The original 4-ft. wood pipe feeds two welded steel pipes having average internal diameters of 950, 850, and 750 millimetres in three equal sections.

The 5-ft. oregon wood pipe feeds a 49-in. karri wood pipe which conveys the water to 400-ft. head. This pipe then feeds a 24-in. steel pipe, the upper half of which is rivetted and the lower half welded, and a welded steel pipe the upper and lower halves of which are 51 in. and 48 in. outside diam. respectively.

The two 5-ft. karri wood pipes each feed 54-in. wood pipes which at 200-ft. head in turn feed 54 in. rivetted steel pipes.

At 400-ft. head one of the 54-in. pipes feeds a 51-in. and 48-in. outside diam. welded pipe and the other feeds a lock-bar pipe. This latter pipe is 51 in. outside diam. at top, and branches into two 36-in. outside diam. pipes.

All the seven steel pipes connect at 1120-ft. head to a bus pipe 900 millimetres diam. running parallel to and just outside the power house. Valves are provided for sectionalizing the bus pipe and for cutting out any one pipe line, so that a considerable flexibility of supply is available.

At the lower end of each wood pipe an automatic throttle valve is provided which closes automatically when the velocity of water due to a breakdown stream exceeds a predetermined quantity.

The wood pipes are supported on wooden bed-logs adzed to shape of the pipe and laid accurately to grade. No anchorage is provided except at dead ends.

Steel pipes are supported on concrete piers, and are encased in large concrete anchor blocks at bends. This is necessary on account of the large thrusts developed by water pressure and temperature stresses. Expansion joints of the sliding stuffing-box type are provided between anchorages. The rivetted and lock-bar pipes are rivetted at circumferential joints, but welded pipes are bolted together by means of loose flanges.

Power House.

Water is supplied to the turbines at 1120-ft. head approximately through nozzles, and regulation is effected by means of needle valves and deflectors controlled by oil pressure governors. The turbines are in two sizes—5000 b.h.p. and 8000 b.h.p.—two of the former and seven of the latter, all 375 r.p.m.

Each governor is driven by belt from the main shaft, and, in addition to operating the needle valve, also operates a deflector, which deflects the water-jet from the wheel. This deflection can be carried out suddenly, practically without pressure-rise in the pipes, and the needle can then move slowly to the new position required by the load. Dangerous surges in the pipes are thus avoided.

The 5000 h.p. wheels have single nozzles, and the 8000 h.p. wheels have two nozzles each. The wheels consist of cast-steel buckets bolted to discs keyed to the shafts. The turbines are direct connected to three-phase alternators, which are of the revolving field, totally-enclosed type, and generate at 6600 volts and 50 cycles. A hot and a cold air-duct are provided in the foundations, and air is drawn into the machines by fan vanes bolted to the rotors.

Bearings of both turbines and generators are ring lubricated, the oil of the turbine bearings being externally water-cooled. Both ends of all windings are brought out, and the neutral points are earthed through current transformers by disconnecting switches. The machines are protected by out-of-balance relays.

Excitation is provided by two 120-kw. turbine-driven exciters running at 700 r.p.m. and two motor generator sets running at 580 r.p.m. These machines also supply current for a 30-ton overhead crane and for charging a 300-ampere hour battery. A reinforced concrete switch gallery occupies portion of the eastern side of the turbine room. The lower floor on the level of the turbine floor accommodates the main and auxiliary busbars, each bar being contained in a separate concrete compartment running the length of the gallery, and also the necessary potential and current transformers for operating the instruments in the control room. Link switches are provided by means of which any generator or bank of transformers can be connected to either set of bars.

On the upper floor of the gallery are placed the control room and twenty-one 6600-volt remote control oil switches.

Each of the nine generators and of the seven banks of transformers is connected to the busbars through one of these oil switches. The main busbars are divided into three sections by two of these switches, and an alternative connection made through 4 % reactors situated in the basement below. Another oil switch serves for paralleling the busbars, and the two remaining oil switches control the motor-driven exciters and the auxiliary circuits respectively. The control room contains a control board of the bench type. Behind the bench board are vertical boards controlling all auxiliary D.C. circuits and accommodating the temperature indicating gear for generators and transformers. Above the bench board are placed all indicating instruments in connection with the main circuits.

Electrically-operated remote control is provided for all oil switches and generator field rheostats, which latter are situated in the basement below the gallery. The exciter field rheostats, D.C. circuit breakers, and governors are also remote controlled.

The generating machinery and controls are all housed in a steel frame building 264 ft. long by 58 ft. wide, covered with galvanized corrugated iron and supported on concrete foundations. The foundations consist of massive blocks under the various machines, piers under the building columns, and slabs, walls, and arches over the intervening spaces. Three tunnels run the length of the building, one forming the wheel pit which discharges the water from the turbines to the tail race and thence to the River Ouse, and the other two forming the hot and cold air-ducts respectively.

Fibre ducts are provided for all cables, and are encased in the concrete forming the floor.

All high-tension transformers and gear are of outdoor type, and were erected in the open adjoining the power house.

Seven banks, each of three 6600/88000, volt 2333 K.V.A. oil-insulated, water-cooled transformers, are placed on rails on concrete runways. These transformers are delta-connected on the low tension side and star-connected on the high. Two spare transformers are provided, and can be substituted for any other transformers by means of a track and truck.

Oil and water piping systems are installed, the former for drawing off oil and the latter for cooling water. The transformers are connected to the high tension busbars through hand-operated, disconnecting switches and choke coils, and thence to the transmission circuits through remote controlled 88,000-volt oil switches. The high-tension busbars are sectionalized by disconnecting switches so that any one circuit can be isolated and connected to a bank of transformers, and through the auxiliary busbar to a generator for testing purposes.

At each end of the 88,000-volt bus there is a set of aluminium cell arresters connected through a disconnecting switch and horn gap.

The insulators supporting the 88,000 volt buses, choke coils, disconnecting switches, and connections are attached to three lattice-steel structures running parallel to the line of transformers.

TRANSMISSION LINES.

Power is transmitted to Hobart and Launceston by means of three transmission lines—two double-circuit lines run to Hobart and one single-circuit line runs to Launceston. In addition, a single-circuit line runs from Hobart to Electrona.

The first two circuits erected were from Waddamana to Hobart, crossing the Derwent River at Bridgewater, and of total length

65 miles. The cables were of 7/.097 copper strands, and were suspended by 5 disc insulators from suspension towers and strained by 6 disc insulators at strain towers.

The third and fourth circuits consisted of 19/.083 copper stranded cable suspended by 4 disc insulators and strained by 5 disc insulators. These circuits cross the Derwent River at Risdon, the total length from Waddamana to Risdon being 62¼ miles.

The Bridgewater crossing of 1180 ft. span was carried out with 0.169 sq. in. silicon bronze conductors, and necessitated one 160-ft. tower.

The Risdon crossing of 1872 ft. span was carried out with 37/.091 copper weld stranded cables, suspended between towers 136 ft. and 171 ft. high.

The line to Electrona is switched at New Town from Nos. 1 and 2 circuits, and the cables are similar to Nos. 1 and 2 circuits.

The line to Launceston is 53 miles long, and consists of 7/.097 copper cables suspended by 4 disc insulators and strained by 5 disc insulators.

With the exception of the Electrona line, all the tower structures are of steel. The towers on one line to Hobart are galvanized, but on the other lines they are painted, with the exception of the cross arms, which are galvanized. All connections are bolted. The towers on the Electrona line are of wood, with bolted connections.

All lines are protected by a galvanized-steel ground-wire earthed at each structure.

SUB-STATIONS.

Sub-stations have been constructed at Risdon, Electrona, Launceston, and Bridgewater.

Risdon sub-station is fed by 4 circuits from Waddamana, and is equipped with 4 banks of transformers, as follows :—

(a) Two banks each of three 5000 K.V.A. 88,000/11,000-volt transformers.

(b) Two banks each of three 1333 K.V.A. 88,000/6600-volt transformers.

Provision has been made for addition of two more banks of transformers, giving 30,000 K.V.A. extra capacity.

The general lay-out and design are similar to the outdoor plant at Waddamana. The low-tension apparatus is housed in a brick structure, the busbars being enclosed in a reinforced concrete structure similar to the switch gallery at Waddamana.

The control board is of the vertical type, and all oil switches are remote controlled.

The 88,000-volt bus is divided into two sections by a disconnecting switch, and one bank of both the 11,000 and 6600-volt transformers is connected to each half.

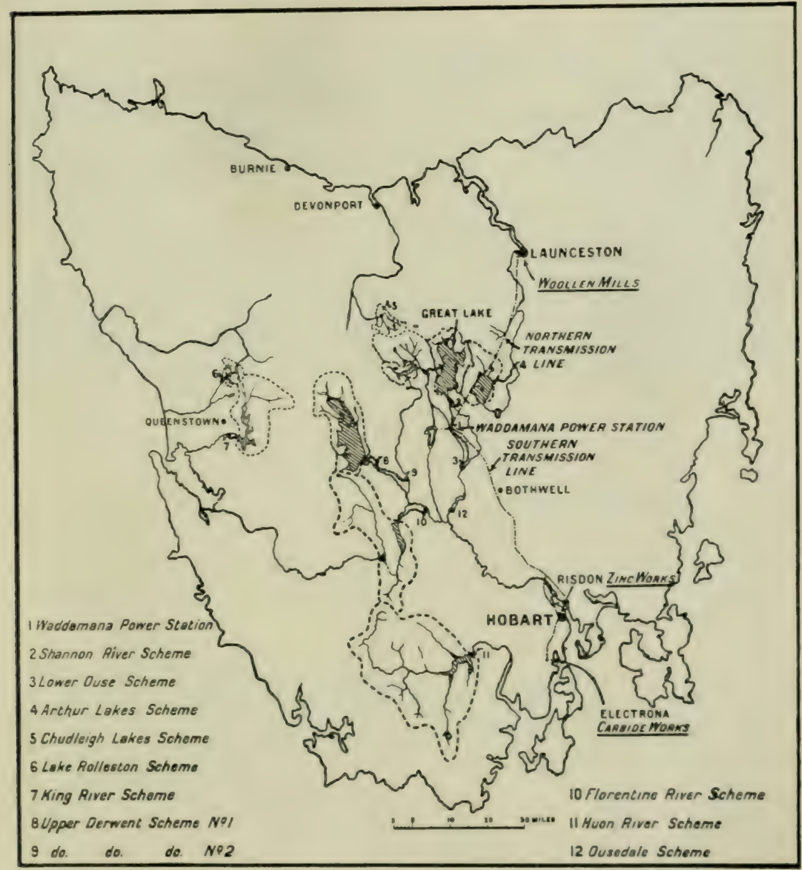

FIG. 1.

MAP OF TASMANIA SHOWING EXTENT OF HYDRO-ELECTRIC SCHEME.

The Launceston sub-station is equipped with one bank of three 1666 K.V.A. 88,000/6600-volt transformers and one spare transformer. These transformers were part of the original installation, and are of the indoor type. They are housed in a brick building together with the 6600-volt apparatus, but all 88,000-volt gear is placed outdoor.

88,000-volt air-break switches were installed in place of the usual oil switches.

The Electrona sub-station is of outdoor type, and is equipped with one bank of three 1333 K.V.A. 88,000/6600-volt transformers and the usual oil switches and electrolytic arresters.

The Bridgewater sub-station is also of outdoor type, and is equipped with one three-phase 1000 K.V.A. 88,000/11,000-volt transformer, with 88,000-volt air-break switches. The 11,000-volt switches are housed.

FUTURE EXTENSIONS.

It is proposed to run transmission lines to Huon district and along the North-West Coast to Wynyard, serving the different municipalities all along the routes. Other lines will be constructed as the loads warrant same.

The whole of the works described have been, since the inception, continuously under the control of the present Chief Engineer and General Manager, Mr. J. H. Butters, M.I.E.E., M.I.E.Aust.

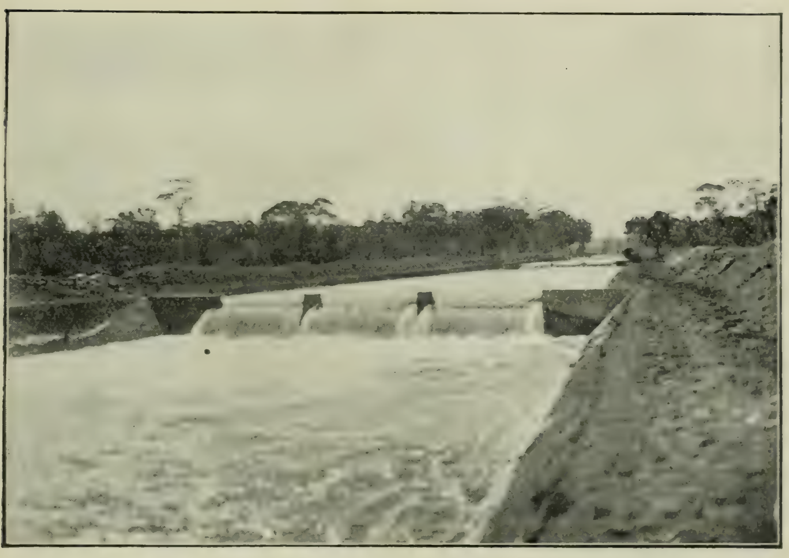

FIG. 2.
WADDAMANA CANAL—SHOWING DROPS.

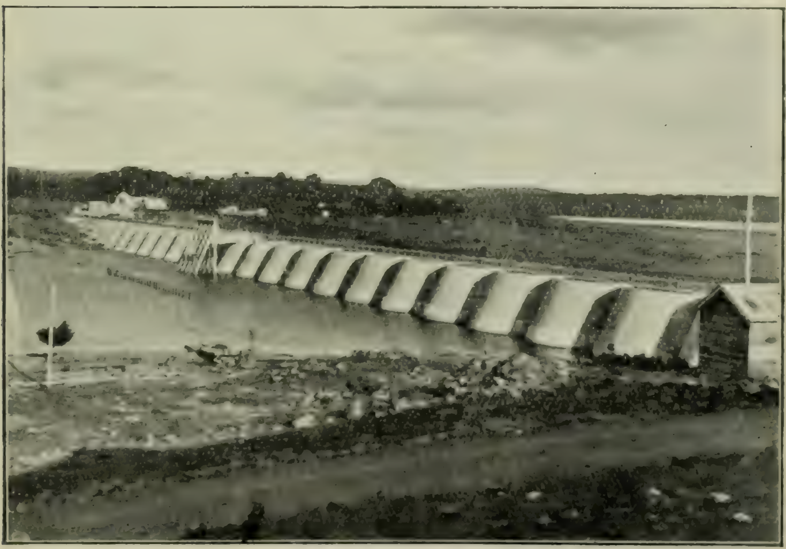

FIG. 3.
NEW MULTIPLE ARCH DAM.

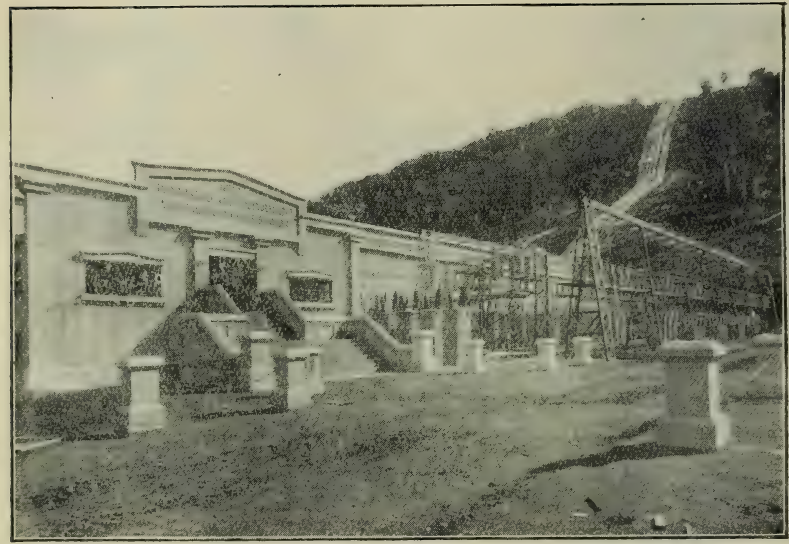

FIG. 4.
WADDAMANA POWER STATION.

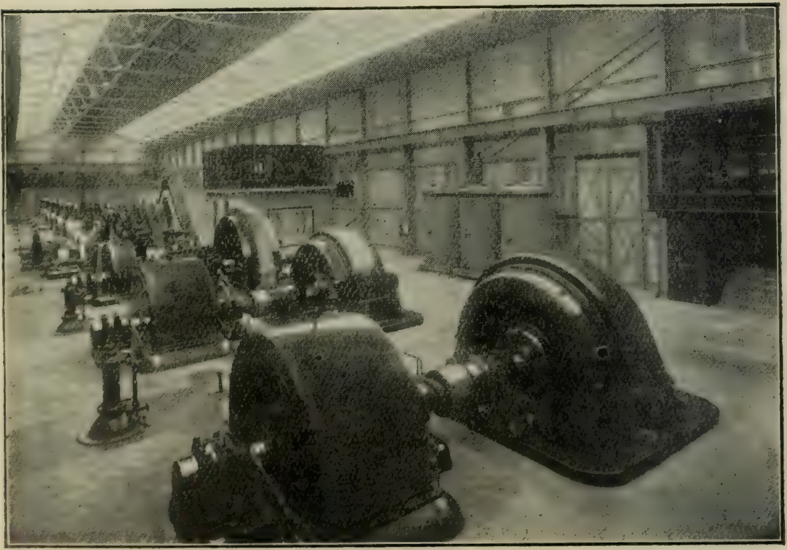

FIG. 5.
INTERIOR OF WADDAMANA POWER STATION.

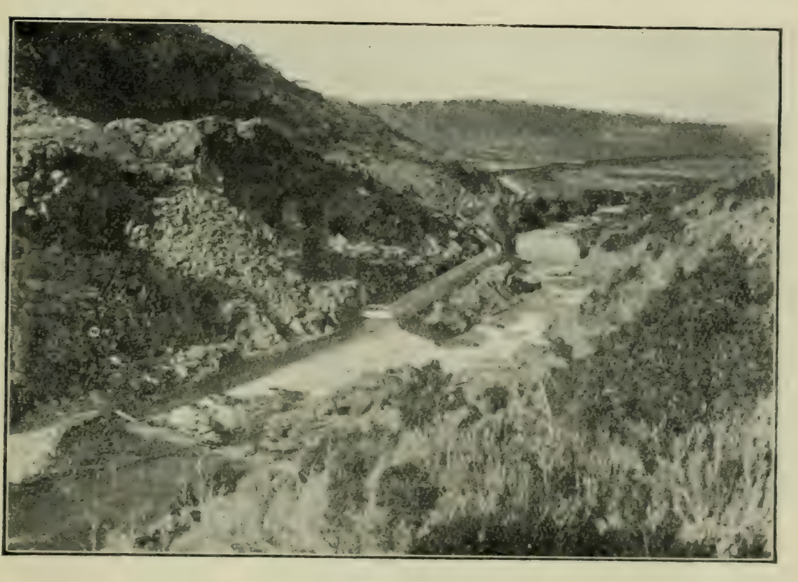

FIG. 6.
OUSE DIVERSION WEIR.

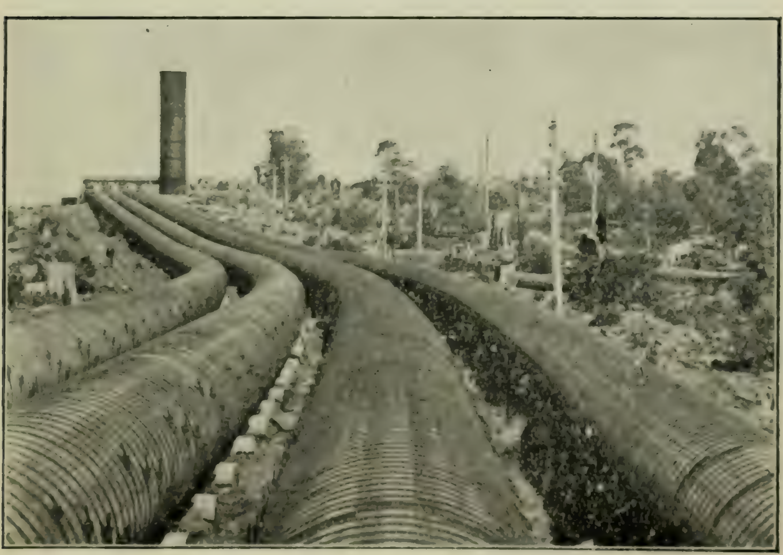

FIG. 7.
WOOD STAVE PIPE LINE WITH STAND PIPE.

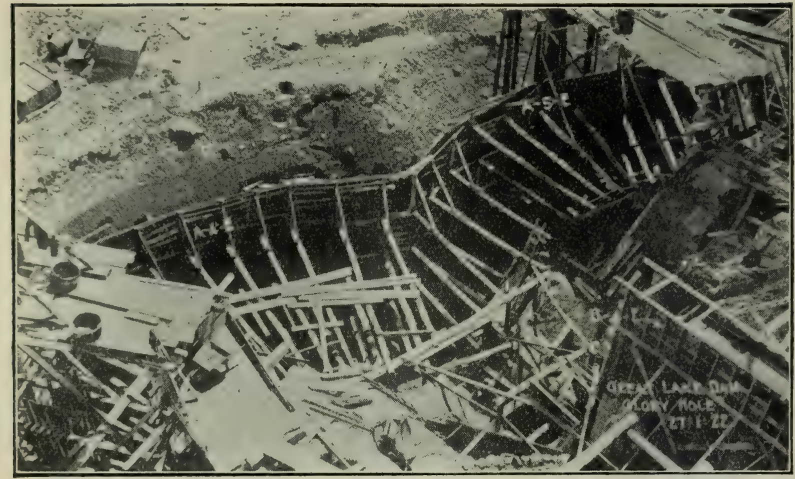

FIG. 8.
GREAT LAKE DAM. 27/1/22.

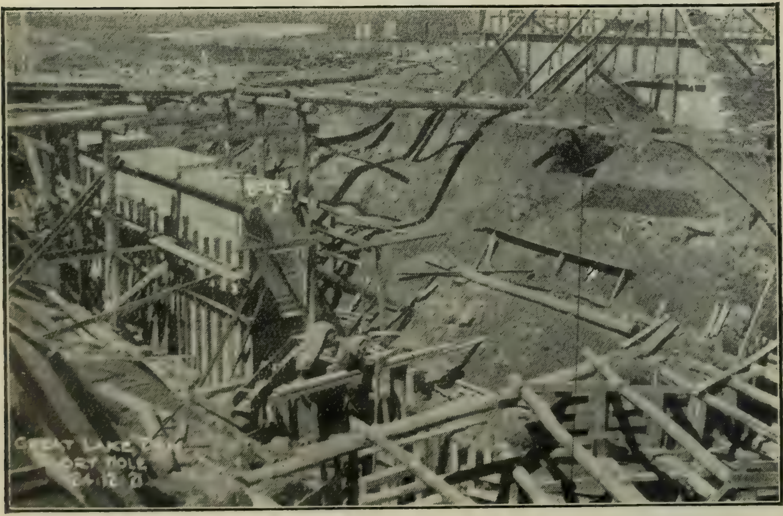

FIG. 9.
GREAT LAKE DAM. 24/12/21.

THE ACTION OF LIGHT ON CINNABAR.

BY W. H. CROPP.

THE object of this note is to set forth some particulars of the effect of light on cinnabar occurring in a deposit of this mineral at Puhipuhi, in the North Auckland peninsula of New Zealand. This deposit is contained within a gently-inclined sheet of siliceous sinter forming a hillock called Mount Mitchell, and is particularly notable in that it is an instance of the simultaneous deposition of cinnabar and silica on a large scale. The mercury-bearing material varies in character from semi-porous sinter to dense translucent chalcedony, of which the former is stained to various shades of pink, the latter to brilliant reds and vermilion. On exposure to daylight—and particularly to sunlight—these colours change to corresponding shades of translucent bluish-grey. As a direct result of this change the Mount Mitchell deposit remained unknown until some ten years ago, for, while prospectors early discovered cinnabar in the adjacent clay and in a neighbouring stream bed, none of them recognized this mineral in the bluish chalcedony strewing portions of the surface of the Mount. The writer has carried out a series of experiments designed to indicate the cause and nature of this change, and to form a basis for the elaboration of more delicate experiments which would yield positive information. Circumstances, however, have so far combined to prevent the continuation of the research, and, as there seems little prospect of the writer being able to do more work on the subject, it seems well to publish the data so far obtained.

Prior to describing this colour change, the writer would point out the desirability of restricting the meaning of the term "cinnabar" to the naturally occurring red sulphide of mercury; this would eliminate the ambiguities which arise from its indiscriminate application, in chemical literature, to both natural and artificial sulphides. Further the term "cinnabarite" having long since fallen into disuse, there appears to be no good reason for retaining the companion term "metacinnabarite" when reduction to "metacinnabar" would preserve the meaning in a less clumsy word.

NATURE OF THE ORE.

Viewed in thin section under the microscope, the cinnabar-bearing material shows a ground-mass of glassy silica which, because of strain and incipient crystallization, is not completely isotropic. Scattered through this are more or less circular masses, each of which is composed of a nucleus of minute angular pieces of quartz (?) surrounded by a radial growth of fibrous silica. The fibres exhibit straight extinction, and, with the angular silica, show a higher birefringence colour than the ground-mass. In some portions the fibrous member is absent and the angular silica merges into the ground-mass; in others the angular member is wanting, and the fibres appear to be filling cavities in the ground-mass.

Cinnabar, in a state of extreme subdivision, is distributed irregularly through the ground-mass to which it is confined and with which it has been deposited simultaneously. Here and there the cinnabar takes the form of isolated floccules suspended in the clear chalcedony, but most of it shows as a red cloud which cannot be resolved by the highest power of a good microscope. In but few parts are the minute component grains distinguishable. The outer portions of the cloud die out imperceptibly in the colourless ground-mass, but one edge—that at which the colour is densest and therefore the concentration of cinnabar greatest—may terminate abruptly against what appears to have been the wall of a transient cavity. The ground-mass in the vicinity of some of the patches of cinnabar is stained light brown with ferric oxide.

THE EFFECT OF LIGHT.

When a freshly-broken surface of the ore is exposed to direct sunlight the colour rapidly changes from the delicate pinks and brilliant reds to shades of translucent bluish-grey of corresponding depths. The change is most rapid and greatest in those specimens which consist of semi-transparent, lustrous chalcedony fairly evenly stained pink to scarlet; in these the cinnabar is extremely fine and in cloud form. Areas wherein the cinnabar is highly concentrated change much more slowly, as do certain specimens of light pink, semi-porous sinter in which some of the colour is doubtless due to iron. At first merely superficial, this colour change extends to appreciable depths on prolonged exposure to sunlight, but the change induced by indirect or diffused sunlight is barely noticeable even for exposures of many months ; and apparently the change is permanent, for altered specimens show no evidence of a return to the original red colour after retention in darkness for four months.

Rate of Change.—For the same intensity of sunlight the rate of change is directly proportional to the density of the colour—that is, to the concentration of cinnabar. A specimen of translucent pink chalcedony exposed to bright summer sunlight (lat. 36–40 south) showed a recognizable change at the end of the second day, and in seven days the alteration to a light blue-grey was complete. A piece of scarlet chalcedony was noticeably dull on the second day and had a film of bluish-grey, through which the red was quite visible, at the end of the seventh ; in fifteen days the surface colour was lead-grey, with only a faint suggestion of red. But it required six weeks of exposure to dull, and three months (with only an occasional wet day) to change to grey, the surface of a small area in which the cinnabar was so concentrated as to appear pure. To state the case in another way, the rate of change is directly proportional to the state of subdivision and dissemination of the mineral.

Depth of Change.—The depth to which the change extends is governed by the concentration of the cinnabar, the amount of light transmitted to it by the enclosing silica, and the length of exposure. A specimen exposed to summer sunlight from 11th October, 1922, to 7th May, 1923—seven months—and then broken to provide a fresh vertical section, showed alteration extending from 1/16 to 7/16 in., with an average penetration of 3/16 in. Of these, the first figure is for scarlet portions, whilst the second measures the penetration in light pink silica. In the case of the small area of highly-concentrated cinnabar referred to in the previous paragraph, the change was only 1/64 to 1/32 in. deep, though the pink silica enclosing this rich patch was altered to a vertical depth of 5/16 in. Ore that has lain exposed on the surface of Mount Mitchell since its deposition in Pliocene times is almost bleached for several inches or more in depth, but this amount of change is subject to the area having been clothed with forest until swept bare by recent fires.

Extent of Change.—It is yet doubtful to what extent this phenomenon can continue. After seven months of exposure the specimen just described is a pronounced lead-grey, and the writer cannot say that the colour is lighter or darker in shade than what it was at the end of the first month ; but T. A. Black,* who has now resided on the Mount for several years, asserts that the grey colour slowly disappears and leaves the silica white on the surface. Certainly the cinnabar-bearing portion of the Mitchell deposit is not readily distinguishable, by superficial examination, from the surrounding barren portion, for the former rarely shows more than an occasional tinge of blue on the surface ; the rock must be broken to expose the scarlet colour. Whilst the greater part of this superficial bleaching may be ascribed to the prolonged action of light, some of it is doubtless due to the heat of recent forest fires.

THE NATURE OF THE CHANGE.

It is well known that certain salts of mercury are sensitive to light and decompose under the action of the ultra-violet rays. The references to mercuric sulphide are many, the statement to be found in the larger text-books on chemistry being that cinnabar (with the term applied indiscriminately to both natural and artificial sulphides) darkens on exposure to light in consequence of a decomposition into free mercury and sulphur. Although this statement is probably a true interpretation of the reaction, recent research does not appear to have definitely established it. In fact, it appears to be founded principally on an early paper by Alsberg † on the manufacture of vermilion, wherein the author, noting the colour change and deciding that it is confined to the surface, is nevertheless uncertain whether it is due to decomposition or simply to a conversion of the crystalline into the amorphous modification. Heumann ‡ examined the effect of light on mercuric sulphide prepared in different ways, and states that the change took place only on the extreme surface ; he was hence unable to obtain the product in quantity sufficient for analysis. He inclined to the view that the change was allotropic, for microscopic examination failed to reveal freed mercury.

The subject was referred to more recently by Allen and Crenshaw while describing the results of an investigation into the formation of mineral sulphides.* Though their remarks deal more with the action of heat than with that of light, they yet have a direct bearing on the problem :—

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* T. A. Black, "Quicksilver in New Zealand," *Chem. Eng. and Min. Rev.* 5/10/22.

† M. Alsberg, "Observations on the Manufacture of Vermilion," *Chem. News*, vol. xxiii., pp. 73–74, 1871.

‡ Heumann, *Ber. Deut. Chem. Gesell.*, Jahrg. 7, p. 750, 1874.

"In the first place, cinnabar turns black as the temperature rises. This is not due to any transformation, but merely to a variation in the absorption of light with changing temperature, for, after it has been heated to 325° C. for many hours, cinnabar quickly regains its colour on cooling. If, however, the temperature is carried to 445°, or perhaps to a lower point, the colour remains permanently black. This was first interpreted to mean a transformation into metacinnabar, but a microscopic examination failed to disclose anything but cinnabar, while by grinding in a mortar the black colour was found to be due merely to a thin coating."

They conclude that the change is not due to the formation of the black allotrope metacinnabar, but do not suggest that it is due to decomposition.

Opinions thus divide between chemical decomposition and molecular rearrangement, with, apparently, little positive evidence for either.

EXPERIMENTAL WORK ON MITCHELL ORE.

Four simple experiments were carried out to show the cause and nature of the change induced by the action of sunlight on Mitchell cinnabar ore, and to indicate the direction that more delicate research should follow. A scarlet specimen was crushed to 80 mesh to increase the rate of change and to give escape to any elements if released. Analysis showed it to contain only 0.06 % of mercury—a much smaller quantity than the brilliance of the colour would suggest. It should be noted that this specimen represents only such ore as is most rapidly affected by light; the more valuable but less spectacular portions of the ore-body frequently give assays of 1% or more.

(1) Some of the crushed ore, with a chip of the same material to act as indicator, was exposed for a fortnight to the heat rays of the sunlight, the ultra-violet and visible light rays being filtered out.

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* E. T. Allen and J. L. Crenshaw, "The Sulphides of Zinc, Cadmium, and Mercury: their Crystalline Forms and Genetic Conditions," *Am. Jour. Sc.* (4th series), vol. xxxiv., p. 379, 1912.

It was thus subjected to temperatures similar to those ruling under natural conditions at the Mount—15° to 42° C.—without exposure to the chemically active members of the sun's rays. No change in the indicator, nor any loss of mercury from the ore, was discernible.

(2) A second portion, with indicator chip, was exposed for a total of 90 hours of strong sunlight, a fresh surface being presented each day. Although the indicator was completely altered, careful analysis showed no loss of mercury from the ore.

(3) A thin section, minus cover-glass, was exposed to sunlight for three months and then examined under a petrological microscope. No free mercury, nor any change other than that of colour, could be distinguished even with a magnification of 290 diameters.

(4) One hundred grammes of crushed ore were placed in a stoppered flask in which a weighed piece of pure silver foil 8 sq. in. in area was suspended ; a calcium chloride drying tube was included to maintain dryness within the flask. A second flask containing a globule of mercury and a suspended weighed piece of silver foil, and a test tube containing an indicator chip of ore, completed the details of the fourth experiment. The three pieces of apparatus were exposed for a total of 60 hours sunlight, both pieces of foil being weighed periodically. During this time the indicator chip underwent a complete change in colour and the foil above the ore attained constant weight ; further, the bright colour of the silver became tarnished by a distinctly brown film, perhaps 1/16 in. in width, affecting the lower edge of the foil. The silver suspended above the globule of mercury assumed a notable change in lustre and gained 0.0011 grm. in weight, indicating the rate of evaporation of mercury under the conditions of the experiment. The foil in the ore-flask took on a similar lustre (though much less pronounced), and gained 0.0056 grm. in weight.

It is conceivable that the brown film was silver sulphide formed by the liberation of sulphur from the ore, but it would be a matter of practical difficulty to prove this. It is further conceivable that part of the gain in weight noted for the foil suspended above the ore was due to mercury liberated from the cinnabar, and an attempt was made to prove this by chemical means. The foil was dissolved and all metals in the solution precipitated as sulphides ; from this precipitate the silver was extracted by prolonged digestion with frequent changes of dilute nitric acid. This left a small dark residue from which a neutral solution of metallic chlorides was subsequently obtained, care being taken to ensure, within the limits of the reactions, the absence of silver from such solution. When this was evaporated to dryness in a small copper-foil dish, and the bottom thereof polished with the finger-tip, there were formed three small silver-white areas, to all appearances identical with amalgam spots purposely produced on another piece of foil. But, probably through faulty manipulation, an attempt to confirm the presence of mercury in these areas by the formation of the scarlet mercuric iodide failed.*

Though no more conclusive in result than parallel research by other observers, these experiments lead the writer to the opinion that the colour change resulting from the action of sunlight on Mount Mitchell ore is due to decomposition of the cinnabar into mercury and sulphur by the ultra-violet rays, the blue-grey colour so produced being that of the freed mercury; and because the chalcedony enclosing the cinnabar does not permit the freed mercury to escape, the colour change is cumulative up to the point of complete decomposition of the cinnabar, and the change is permanent. Confirmation or rejection of this thesis would be possible after refined research along the lines of the fourth experiment, using gold foil and examining this by the spectroscope † for the presence of added mercury.

SIMILAR OCCURRENCES ELSEWHERE.

There appears to be no record in mineralogical literature of any other deposit of mercury ore comparable in nature and behaviour to that of Mount Mitchell. The colour of cinnabar is given as vermilion, lead-grey, and (in rare cases) almost black, but there is no mention of any change resulting from exposure to light. What appears to be the sole reference to the phenomenon is the one occurrence noted by Bradley in his description of the Californian cinnabar deposits.‡

In making the general statement that the red sulphide of mercury (natural or artificial ?) darkens under the action of light, he appends a footnote to the effect that he has certain specimens from the Oat Hill mine, Napa Co., Cal., which are affected in a similar manner; but what appears to be the exception at Oat Hill is the rule at Mount Mitchell. Alluvial pellets of finely crystalline cinnabar present in quantity in a swamp near the Mitchell deposit, when finely ground to produce a fresh surface, show a distinct darkening in colour on exposure to sunlight; but the change effected by four months of exposure did not exceed film thickness. Incoherent, finely-crystalline cinnabar occurring in two adjacent claims is said to darken in a like manner, though of this the writer has no personal experience.

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* See Treadwell and Hall, "Analytical Chemistry," 4th Eng. ed., p. 203, 1916.

† Treadwell and Hall, *op. cit.*, p. 204, give suitable details. See also K. C. Browning, "The Detection of Traces of Mercury Salts for Toxilogical Purposes," *Jour. Chem. Soc.*, vol. cxi., pp. 236–240, 1917.

‡ W. W. Bradley, Bull. No. 78 Cal. State Min. Bur., p. 208, 1918.

COMPARATIVE COST OF ROCK DRILLING UNDERGROUND.

By Thos. G. Hanton.

During a period extending over about the last two and a half years, the performances of various types of rock drills in use at the mining department of the Zinc Corporation have been recorded as closely as possible.

The main objective was the collection of sufficient data regarding internal water-feed hammer drills, when used for the same class of work as the older type reciprocating drill, to enable a decision to be formed on the following points :—

(1) Whether internal water-feed hammer drills could efficiently perform the work previously carried out by reciprocating drills without excessive repair cost to the former.

(2) If so, was it good policy commercially to eliminate the reciprocating type completely, writing these off together with the accompanying spares, accessories, and solid drill steel, and replacing with internal water-feed hammer drills, with lighter fittings, and the requisite amount of hollow drill steel.

(3) What were the best types and makes of hammer drill for different classes of mining work ?

(4) What was the economic aspect of the introduction of the principle of internal water feed to every type of machine throughout the mine ?

Prior to the strike of workmen in May, 1919, Ingersoll-Rand E47 3¼-in. reciprocating drills had been used for stoping and for the major portion of driving, cross-cutting, and similar work. The accompanying columns and fittings were all 4½ in. The drill steel used was 1½ in. and 1¼ in. solid octagon.

Up to 1916 these machines were also used for winzing, after which time jackhamers were more successfully employed. These latter were of the usual types, varying in weight from 40 to 46 lb.—Ingersoll BCR 430, Holman No. 2 spool valve, Sullivan DP33, Waugh Clipper 55. These machines were all dry head ; the drill steel was ⅞-in. hollow hexagon, shanked.

For popping, these jackhamers quickly replaced the Waugh 22P (weight 33½ lb.), Sullivan DB19, and similar hand-rotated poppers formerly employed for the purpose. The miners preferred the jackhamer, although it was somewhat heavier, on account of its automatic rotation and faster boring speed.

For "rising," various telescopic air-feed machines, such as Waugh 16V, Ingersoll CC11, and Sullivan DA21, were used. With these the application of water was by an external jet. The drill steel was 1⅛-in. solid cruciform.

The first internal water-feed machine designed for the same class of work as the 3¼-in. reciprocating drill arrived on the mine in 1916. Its weight was 147 lb. This was followed by three others of different manufacture, but was the only one which proved satisfactory. Its record will be given later. The others did very little work, and, after a further trial and thorough examination, were discarded.

It should be borne in mind that these machines, when introduced, were quite a new feature. There was, consequently, a lack of knowledge on the part of the miners, and the importance of care in the preparation of the hollow drill steel was not properly appreciated. These earlier machines, therefore, worked under rather adverse conditions. With hammer drills now in fairly general use, much greater regard is given to those features essential to their successful operation, and the care and treatment accorded the machines is now very much better.

The period covered by the present record may be divided into two portions. During the first of these stoping operations were suspended, and the only underground work in progress consisted of the driving, cross-cutting, winzing, rising, and other excavation required for the fulfilment of the ventilation scheme decided upon for the mine.

In the second portion (extending from January, 1923) full stoping operations have been in progress, with only a small amount of ventilation and development work.

It was during the first portion that most of the rock-drill trials were made. The results obtained in various classes of work indicated the types and makes of machines most suitable for adoption on the resumption of production. Reservations were, of course, necessary in some instances until further particulars under actual stoping conditions were secured.

Before setting out the performances of various machines, it may be stated that the boring time mentioned includes all changes of steel, shifting to different positions of bar and arm when machine is mounted, and very frequently includes the time occupied in rigging the bar. It is, therefore, not actual drilling time, but time while the machine is in use, and thus takes into account the ease in handling. Actual drilling time may be only from 40 % to 50 % of this time, according to men and conditions.

In the following record, average grade ore may be taken as sulphide ore containing about 14.5 % Pb and 9 % Zn. Anything higher than this is termed good ore, whilst low-grade ore may be taken as containing from about 3 % to 4 % Pb. Country consisting of a good proportion of fine-grained quartzitic material comes under the heading of hard country. The hardest country in the mine contains a considerable amount of garnet sandstone, mostly very fine-grained. In down holes this is especially troublesome, as it does not sludge well, and the cuttings remain at the bottom of the hole. Frequent blowing-out has been found the best remedy. The granular rhodonite common in the mines on the centre and north end of the field is not present.

The cost of repairs given does not include any wages for fitting. It includes only new spares and the cost of any parts made or repaired at the workshop. One rock-drill fitter is constantly employed. His duties include the making up and mending of all hoses and repairs to all fittings and accessories. The cost of spares is based on the actual price paid at the time. No attempt has been made to take into account the variations from time to time. The same applies to the first cost of the machines.

Compressed air is delivered from the power plant to the receiver, from which the mine supply is drawn at an average pressure of 80 lb. per sq. in. The air pressure at the working places varies from 73 lb. per sq. in. to 78 lbs. per sq. in. according to their position.

RESULTS FROM DEVELOPMENT WORK.

DRIVING.

(*a*) Air-feed hammer drill, hand rotated, in which water was admitted through the front end of the machine. The weight is 86 lb. The weight of the piston is about 4½ lb.

The drive was 8 ft. x 8 ft., the material fairly hard, low-grade quartzitic ore. After 232 ft. were bored the trial was discontinued. The physical strain on the operator when rotating the drill in holes more or less horizontal rendered it unsuitable for Zinc Corporation conditions. With an air-feed machine, the pressure on the drill bit in holes vertically upwards is the difference between the pressure on the pole and the weight of that part of the machine which is fed forward. In vertically down holes it is the sum of these two. In intermediate positions it varies between the above limits.

Trouble was experienced with the anvil block packing, which quickly wore out, permitting water to get into the cylinder, and thus causing fogging. This trouble could be obviated to some extent. With this method of water admission there will probably always be a certain amount of leakage between the anvil block and the drill steel. The makers of a machine similar to this state that the maximum water pressure should not exceed 30 lb. per sq. in. Another disadvantage is that the steel used (1-in. hollow hexagon) has plain ends, and thus there is difficulty in withdrawing a tight drill from the hole. This, of course, is an advantage in the sharpening shop, but is not so underground.

(*b*) Internal water-feed hammer drill, with automatic rotation independent of the piston hammer. This machine is valveless. Further particulars are :—

Weight of machine complete on guide shell, 168 lb. ; piston diam., 3½ in. ; travel, 3½ in. ; weight, 12 lb. 3 oz.

Details supplied from outside sources :—

Blows at 60 lb. per sq. in. air pressure, 1500 per min.
,, ,, 80 lb. ,, ,, 1700 ,,
,, ,, 100 lb. ,, ,, 1700-1800 per min.

Foot lbs. of blow at 60 lb. per sq. in. air pressure, 44.
,, ,, ,, 80 lb. ,, ,, 56.
,, ,, ,, 100 lb. ,, ,, 64. to 73.

Inches drilled per min. at 60 lb. per sq. in. air pressure, 6.
Inches drilled per min. at 80 lb. per sq. in. air pressure, 9.5.
Inches drilled per min. at 100 lb. per sq. in. air pressure, 13.6.
Material not stated

Air consumption at 60 lb.—73.4 cub. ft. free air per min.
,, ,, at 80 lb.—99.9 ,, ,, ,,
,, ,, at 100 lb.—120.0 ,, ,, ,,

Results supplied from other outside tests are :—

Blows per min.	1632.
Maximum force of blow ..	500 lb. per sq. in.
Minimum ,, ,, ,, .	400 lb. per sq. in.

The drill steel used was 1¼in. hollow round.

This machine started work on 21st April, 1921. The following is its record :—

	Footage Bored.	Boring rate Feet per hr.	Boring Time hrs	Footage Advanced.	Progressive cost of spares.			Material.
					£	s.	d.	
8 x 8Drive	652	8.2	79	26.5		—		Fairly hard low-grade quartzitic ore.
8 x 8Drive	1992	12.5	159	68.5		—		Fairly good sulphide ore.
7 x 7Xcut	1441	9.2	157	61		—		Medium hard country
8 x 8Drive	1318	7.3	179	98.5		—		Fairly hard ore.
8 x 8Drive (A)	961	10.0	96	40.5	12	19	10	Low-grade quartzitic ore.
8 x 8Xcut	2101	10.8	195	90.5		—		Fairly hard country, 30.5′; good sulphide ore, 60′.
8 x 8Drive	712	12.5	57	29		—		Good sulphide ore.
8 x 8Xcut	2059	10.2	202	65.5		—		Good sulphide ore.
10 x 6Rise (B)	—	—	—	6.5	42	15	8	
8 x 8Xcut	1191	6.5	184	46	53	2	5	Very hard country; garnet sandstone, etc.
8 x 8Drive	1765	6.5	273	63	57	3	0	Very hard ore; much garnet sandstone.
8 x 8Xcut	434	9.6	45	16		—		Fairly hard sulphide ore.
8 x 8Drive	2800	9.1	305	87	71	12	6	Hard sulphide ore.
Totals ..	17426	9.0	1931	712.5	71	12	6	

Summary.

	s.	d.
Repair cost per ft. of heading from start to finish ..	2	0.1
Cost of machine, new, on mine, £126.		
Cost per ft. of heading for 712.5 ft. to cover first cost ..	3	6.6
Total per ft. of heading, first cost plus repairs	5	6.7
Repair cost per ft. bored	0	0.99
Cost per ft. bored to cover first cost	0	1.73
Total per ft. bored, first cost plus repairs	0	2.72

Considering the various stages in the above list :—

	Footage Bored.	Boring Rate per hr. Feet.	Boring Time hours.	Footage Advanced.	Cost of Spares.
					£ s. d.
Start to A ..	6346	9.5	670	309	12 19 10
Start to B ..	11236	10.0	1124	500.5	42 15 8
B to finish ..	6190	7.7	807	212	28 16 10

i.e. —	Repair Cost per ft. heading.	First Cost per ft. heading.	Total per ft. heading.	Repair Cost per ft. bored.	First Cost per ft. bored.	Total per ft. bored.
	s. d.	s. d.	s. d.	s. d.	d.	d.
Start to A	0 10	8 1	8 11	0.49	4.76	5.25
Start to B	1 8.5	5 0.5	6 9	0.91	2.69	3.60
B to finish ..	2 9	—	—	1.12	—	—

The machine was then considered to be so badly worn generally that it would be truer economy to scrap it than purchase the necessary parts to put it into good working order. Its efficiency was low, due principally to wear on the cylinder and piston. It apparently worked quite satisfactorily on the anvil block in the shop, but the force of the blow was not sufficiently great. Various alterations were made, regulating the air supply for piston hammer and rotation When sufficient air was admitted to the rotation gear to obtain a good turning effect, the hammer action was weak, and *vice versâ.*

Considering the increased setting price necessitated by the falling-off in boring speed, and the increasing cost of repairs in the last stage, it would probably have been sound policy to have written off the machine at stage B, after 11,236 ft. of boring and 500.5 ft. of advance of heading.

It may be noted that this drill was in constant use in developmental ends, working two shifts during the greater part of its career. The last 212 ft. of heading was all in hard ground—much of it particularly so. Its life, reduced to a two-shift basis, and in commission constantly. without a break, was sixteen (16) months.

The principle of independent automatic rotation has some attractive features, and probably will be further developed. It appears, however, more suited for soft ground than hard. With independent rotation the drill is turning regardless of the position of the piston hammer, instead of coinciding with a definite position of the latter for each blow. With rotation dependent on the hammer action each blow should, in any ground, have a definite chipping effect, more approaching the old hand-labour style. The action of the bit with independent rotation may be more a rubbing than a chipping one. In soft rock the result is good ; but in hard rock the tendency is to wear down the wings of the drill bit fairly quickly, with resultant slower boring as the bit loses its clearance.

With combined rotation, full air is acting on the piston ; with independent rotation, portion is diverted to the rotation mechanism, and after wear takes place the tendency is for the force of the hammer blow to be weakened.

(*c*) Original axial water-feed machine, using 1¼-in. hollow round steel ; weight 147 lb. ; butterfly valve. Its approximate record is :—

	Footage Bored.	Boring Rate per hr.	Boring Time hrs.	Footage Advanced.	Progressive Cost of Spares.	Material.
					£ s. d.	
8-ft. x 8-ft. drive ..	—	—	—	840	—	Average grade sulphide ore.
8-ft. x 8-ft. Xcut. ..	—	—	—	94	—	Fairly hard country.
Shaft stripping ..	—	—	—	46.5	—	Fairly hard country.
Stoping ..	—	—	—	1642 tons	85 5 1	Average sulphide ore
8-ft. x 8-ft. drive and xcut. ..	1505	9.2	163	54	88 11 9	Hard ore.
Stoping ..	—	—	467	3956 tons	117 2 2	Average sulphide ore

Total.

		s.	d.
Driving and crosscutting	988 ft.		
Shaft stripping	46.5 ,,		
Tons broken stoping	5598		
Repair cost per ft. of heading (988 ft.) for spares, £88 11s. 9d. =		1	9.5
Cost per ft. of heading to cover first cost, £91		1	10.1
Total per ft.		3	7.6

This machine has been taken out of commission, as it is now in such a condition that its upkeep cost is too expensive for regular work. It may be noted in this connection that the repair cost per ton for the last 3956 tons broken is £28 10s. 5d., or 1.7d. per ton, which is well above the average, as will be shown later.

(*d*) An internal water-feed machine of same manufacture as (*c*), but of a later type, superseding (*c*). Weight, 156 lb. Steel used $1\frac{1}{4}$-in. hollow round.

Other particulars supplied are :—

Blows per min.	1332
Maximum force of blow	600 lb. per sq. in.
Minimum force of blow	400 lb. per sq. in.
Maximum length of stroke ..	3 in.
Minimum length of stroke	$2\frac{1}{2}$ to $2\frac{5}{8}$ in.
Piston diam.	$3\frac{1}{4}$ in.
Weight of piston	9 lb. 3 oz.

The machine was sent on trial in August, 1921. Its record is :—

	Footage Bored.	Boring Time hrs.	Boring Rate per hour.	Footage Advanced.	Material.
8-ft. x 8-ft. drive ..	380	31	12.2	16	Low-grade quartzitic ore.
7-ft. x 7-ft. crosscut	586	50	11.7	26.5	Medium hard country
8-ft. x 8-ft. drive ..	1080	103	10.4	50.5	Fairly hard siliceous sulphide ore.
Total ..	2046	184	11.1	93	

Repairs—1 water tube ; cost, 5s. 6d.

This machine was then removed for demonstration purposes elsewhere.

A drill of the same type did 43 ft. of driving in hard ore without repairs, and is now in use stoping. This year another machine of this class (after breaking 2500 tons stoping, with no repairs) has been used in 179 ft. of driving, 8 ft. x 8 ft. in average sulphide ore, for a total repair cost of £8 16s. 2d., or 11.8d. per ft. of drive.

(*e*) An internal water-feed hammer drill; weight, 164 lb.; cylinder diam., 3½ in.; valveless type. Drill steel 1¼-in. hollow round.

The boring speed of this machine compared favourably with similar machines of different manufacture. Its trial was discontinued, as certain features were not regarded as satisfactory. These matters have since been receiving attention from the manufacturers.

WINZING.

For this work it was considered that better results would be obtained with a hand machine, heavier and more powerful than the jack hamers, 40 to 46 lb. weight, previously in use. It also appeared that 1-in. hollow hexagon drill steel would be generally more satisfactory in winzing than the ⅞-in. hollow hexagon drill steel used with the lighter jackhamers. For the latter, using ⅞-in. hollow-hexagon steel, only general figures are available. Although good results were obtained with these machines, the performance of the heavier ones, using 1-in. hollow hexagon steel, appears generally better.

It is essential that the heavier machine should be well cushioned, otherwise the strain on the operator is too great. A blow valve, with which the drill runner may remove sludge from the drill-hole at will is found very convenient.

The general idea was to use these heavier machines when new for winzing, and then, later on, mounted for stoping.

It may be mentioned that since January, 1922, with all winze machines, internal water feed has been used.

(*f*) Jackhamer, internal water feed; weight, 59 lb.; cylinder diam., 2⅝ in.; stroke, 2½ in.; length, 21¼ in.; steel, 1-in hollow hexagon by 4¼-in. shank.

Its record is :—

	Footage Bored	Boring Time hrs.	Boring Rate per hour.	Footage Advanced.	Repair Cost.	Material.
Feet.					£ s. d.	
10 x 6 winze	1894	263	7.2	94	6 9 1	Average grade sulphide ore.
7 x 5 winze	1199	165	7.2	67	7 3 7	Average grade sulphide ore.
7 x 5 winze	586	160	3.7	54.5	3 8 5	Very hard garnet sandstone.
7 x 5 winze	967	143	6.7	52.5	—	Average grade sulphide ore.
7 x 5 winze	1161	167	7.0	100.75	9 18 4	Good sulphide ore.
10 x 6 winze	448	69	6.5	30.75	0 12 6	Fairly hard low-grade ore.
Totals	6255	967	6.46	399.5	27 11 11	

Repair cost per ft of winzing (£27 11s 11d. for 399.5 ft.)	1s. 4.5d. per ft.
Cost per ft. of winzing to cover first cost (£65) ..	3s. 3d. per ft.
Total	4s. 7.5d. per ft.
Repair cost per ft. bored (£27 11s. 11d. for 6255 ft.)	1.05d.
Cost per ft. bored to cover first cost	2.49d.
Total per ft. bored	3.54d.

The above total costs would represent the case if the machine were worn out after the work tabulated. It was, however, mounted, and is now in use stoping.

(f_2) Jackhamer, same type as (*f*).

This machine was used for 74 ft. of 10 x 6 winze in fairly hard country. The repair cost was £1 11s., or 5.03d. per ft. of winze. It is now used mounted for stoping.

(*g*) Jackhamer of English manufacture; internal water feed; weight, 64 lb.; cylinder diam., 2½ in.; stroke, 3¼ in.; length, 24½ in.; steel, 1-in. hollow hexagon; length of shank, 4⅝ in.

This machine was used mounted, as the vibration was too great for use as a hand machine. Its record is :—

	Footage Bored.	Boring Time hrs.	Boring Rate per hour.	Footage Advanced.	Repair Cost. £	s.	d.	Material.
7 x 5-ft. winze	1750	Not recorded		87.5	0	17	6	Very hard, siliceous low-grade ore.
Stripping ..	399	84	4.7	—	0	10	0	Hard quartzitic country
Pump chamber	345	57	6.0	—	7	0	0	Fairly hard country.
Store	266	45	5.9	—		—		Hard country.
Plat A ..	918	217	4.2	—	13	11	0	Hard country.
Plat	176	44	4.0	—	15	7	6	Hard country.
Totals ..	3854				37	6	0	

At "A" on list 3678 ft. were bored for repair cost of £21 18s. 6d. ; equals 1.4d. per ft. bored.

For the 3854 ft. bored the repair cost was £37 6s.=2.3d. per ft. bored.

If the machine had been scrapped after boring 3678 ft., the total cost per ft. bored, first cost, and repairs would have been 6.5d. (first cost £78, plus repairs, £21 18s. 6d.)

It may be noted that the upkeep cost from "A" to the end was very high. The machine is now in use mounted for stoping, but repairs have been costly.

It may be mentioned here, in connection with the 1-in. hollow hexagon steel, that there is not a standard length of shank. The American shank length is 3¼ in. and 4¼ in.—mainly the latter ; the English is 4⅝ in. It is obvious that any more than one shank length on any particular size of steel would lead to confusion in the shop In this case the chucks on the English machines were cut down to accommodate the 4¼ in. length. With 1¼-in. hollow round and ⅞-in. hollow hexagon drill steel the shanks are standard.

RISING.

(*h*) Self-rotated stoper ; internal water feed ; length over all, 61½ in. ; weight, 97 lb. ; cylinder diam., 2¾ in. ; drill steel, ⅞-in. hollow hexagon. plain end.

After boring 141 ft. in hard siliceous country the piston snapped off 6 in. from the striking face. The hole was up 2 ft. 6 in., and the drill was free in the hole when the break occurred. It appeared that the fluting of the piston for rotation purposes, together with the central hole for the water tube, left insufficient cross-sectional area of metal.

In a further trial later, after boring 55 ft. in good sulphide ore, another piston broke in almost exactly the same spot as in the first case, the drill again being quite free in the hole.

It was considered that any machine combining the features of automatic rotation and air feed would need to be of such strong construction that its weight would be excessive.

Trouble was experienced with the mechanical brake provided for retarding the feed when a hole was being collared, or in passing through fitchery ground. This arrangement was not very effective. The makers have now superseded this mechanical brake with an air-actuated contrivance. This has not been tried, as the principle of automatic rotation and air-feed does not appeal, and here there does not appear to be any field for this class of machine.

(*i*) A light stoper (hand-rotated); internal water feed; length, with feed piston drawn in, 48 in.; weight, 66 lb.; cylinder diam., $2\frac{3}{4}$ in.; drill steel, $\frac{7}{8}$-in. hollow hexagon, plain end.

After boring 167 ft. in good sulphide ore the piston cracked. The boring speed was satisfactory. Lightness, however, appears to have been obtained at the expense of durability.

(*j*) Air-feed hammer drill; internal water feed; length, closed, $52\frac{1}{4}$ in.; extended, $74\frac{1}{4}$ in.; weight, 81 lb.; piston diam., $1\frac{3}{8}$ x $2\frac{3}{4}$; stroke, 3 in.; feed, 22 in.; feed diam., $2\frac{1}{4}$ in. drill steel, 1-in. hollow hexagon, plain end.

This machine was previously used dry, and had done a considerable amount of work before conversion to wet. Its record (wet) is :—

Feet	Footage Bored.	Boring time hours.	Boring rate per hour.	Footage Advanced.	Material.
Rise, 7 x 5	234	49	4.7	16.5	Fairly hard country.
Rise, 7 x 5	381	82	4.6	22	Fairly hard ore.
Rise, 7 x 5	483	89	5.4	28.5	Fairly hard country.
Rise, 7 x 5	889	129	6.9	42	Good sulphide ore.
Rise, 7 x 5	123	20	6.0	4	Good sulphide ore.
Totals ..	2110	369	5.7	113	

The repair cost was £11 16s. 4d. = 2s. 1.1d. per ft. of rise.

The repair cost per ft. bored = 1.31d.

The machine is still available for work. This drill showed the advantages of internal water feed as it affects boring speed and the life of the drill steel. When dry, using $1\frac{1}{8}$-in. solid cruciform steel, it was used in the first rise on the above list. Boring was exceedingly slow, in a great measure owing to the difficulty in getting water to the bottom of the drill hole to keep the steel cool, and also to the shape of bit on the cruciform steel.

(*k*) Jackhamer, mounted.

As an experiment, a wet-head jackhamer, weight 40 lb., on a light mounting, and using $\frac{7}{8}$-in. hollow hexagon steel, was used to put up 20 ft. of rise, 7 ft. x 5 ft., in sulphide ore. It was not very successful, and the telescope machine was reverted to. One of the main drawbacks was that the ore was calcitic and short-grained, and thus did not "shoot" well. After boring out, it was almost invariably necessary to rig again once or more to ease some of the holes. With the telescope machine the holes bored each shift could be fired at the end of the shift, and the following holes then placed according to the results obtained. Also, with the mounted machine, after the completion of boring much time was wasted beween the firing rounds.

The foregoing practically completes the results obtained during the first portion of the record.

It may be noted that *a*, in driving, was the only machine in which water admission was not by tube; also that *b*, in driving, was the only one with automatic rotation independent of the piston hammer.

The conclusions arrived at were that for the conditions obtaining on the mine, axial water-feed hammer drills were capable of performing the work previously done with reciprocating drills, without excessive upkeep cost.

Further, taking into account the faster boring speed, lighter weight, and consequent ease in handling of hammer drills as compared with the piston drills (those formerly used on the mine weighed about 270 lb.), and considering carefully the many points involved, it was held to be good policy to discard the older type and replace with hammer drills.

Also, in view of the effectiveness in dust-allaying and reduced wear on drill bits, it was decided that all machines should be equipped with internal water feed.

It was realized that the above changes would entail greater care and closer supervision. This, however, applies to most improvements, and the advantages seemed to warrant the extra trouble involved.

It was considered that four types of hammer drills would be sufficient for the mine requirements, viz. :—

1. A machine similar to the Ingersoll 248, Waugh Turbro, Sullivan DX61, Holman CH2, for driving, crosscutting, and stoping, using $1\frac{1}{4}$-in. hollow round steel.
2. A jackhamer, about 60 lb. weight, for winzing as a hand machine, or for stoping, mounted, using 1-in. hollow hexagon steel.
3. A light, self-rotated popper, lighter, if possible, than the 40-lb. jackhamer, using $\frac{7}{8}$-in. hollow hexagon steel.
4. A telescopic air-feed machine for rising, using 1-in. hollow hexagon steel with a plain end.

There would then be three sizes of steel in the shop.

STOPING PORTION OF PERIOD.

Upon the resumption of productive work in January, 1923, steps were taken to put into effect the foregoing decisions. The necessary $1\frac{1}{4}$-in. and 1-in. hammer drills were purchased. The 40-lb. jackhamers on hand were converted into wet machines, likewise a sufficient number of telescope machines.

Previously, stoping contracts had been on the fathomage system. During suspension of stoping the mine had been equipped for the tonnage system, which for many reasons was more suitable for existing conditions, and operations were resumed on this basis.

There were in the mine about 14,000 tons of ore, broken on the fathomage system, which it was necessary to clean up before starting stoping proper on tonnage. Due allowance has been made for this in the repair costs of stoping drills. Much of it required popping.

From January to date (18/8/23) a total of 107,301 tons of ore has been raised from the mine.

The total cost of repairs for all stoping machines and poppers during this period has been £402 3s. 1d. This is equivalent to 0.90d. per ton on the total tonnage raised, or 1.30d. per ton on this tonnage, less 14,000 tons.

Details of the above costs are as follow:—

Stoping with Reciprocating Drills.—This year a total of 3051 tons was broken with these machines before they were taken completely out of commission. No spares were used, machines being replaced and not repaired.

Stoping with Hammer Drills, Using $1\frac{1}{4}$*-in. Hollow Round Steel.*—

(1) Latest type ("*d*" in driving record), practically all new machines except one, which had been used in 43 ft. of drive.

The tonnage broken with these to date is 40,487. Their total repair cost has been £22 14s. 6d.; this is equivalent to 0.13d. per ton broken.

It may be mentioned that there have not yet been any replacements of the more expensive spares, such as piston hammers, in these machines. When the time arrives for such parts to show wear, the upkeep cost will rise correspondingly. The repair cost per ton when there is an average proportion of older machines will probably be more in the neighbourhood of one penny.

(2) Original axial water-feed machine ("*c*" in driving).

This drill has broken 3956 tons for a repair cost of £28 10s. 5d., or 1.7d. per ton, which, as previously mentioned, is excessive, the machine being almost worn out.

Considering the repair cost for all the above machines, using $1\frac{1}{4}$-in. hollow round steel, £51 4s. 11d., for a total of 44,443 tons, the repair cost per ton for the ore broken with them is 0.28d.

Stoping with Mounted Jackhamers, about 60 *lb. Weight, Using* 1*-in Hollow Hexagon Steel.*—

(1) Jackhamers as "f" and "f_2" in winzing, mounted.

The performance of these two machines prior to stoping has been given. The remainder were new machines.

The tonnage broken with these to date is 26,698. Their total repair cost has been £100 0s. 10d., which is equivalent to 0.90d. per ton broken.

(2) Jackhamers of English manufacture as "*g*" in winzing. One ("*g*") had done the work set out; the remainder were new.

The tonnage broken with these to date is 7991. Their total repair cost has been £39 7s. 4d., which is equivalent to 1.2d. per ton broken.

(3) Lighter jackhamer, using 1-in. hollow hexagon steel; previously used popping with $\frac{7}{8}$-in. hexagon steel.

The tonnage broken with this was 4176. The total repair cost has been £26 1s. 6d., which is equivalent to 1.5d. per ton broken. This is rather higher than it should be, as the machine was somewhat damaged by blasting.

Total tonnage broken with mounted jackhamers, using 1-in. hollow hexagon steel	39,865.
Total repair cost	£165 9s. 8d.
Repair cost per ton	1.02d.

This cost should not rise very much, as there is a due proportion of old jackhamers.

Total tonnage broken with 1¼-in. and 1-in. machines	80,352.
Total repair cost, 1¼-in. and 1-in. machines	£216 4s. 7d.
Repair cost per ton, 1¼-in. and 1-in. machines	0.64d.

The balance of the tonnage raised, after taking into account the amount broken in the mine at the start, comprises ore from taking up bottoms for sill timber with 40-lb. jackhamers, development drives and winzes, and various occasional work not credited to any particular machine.

POPPING.

The jackhamers, 40 to 46 lb. weight, already mentioned as having been converted into wet machines, have been used for popping. The drill steel is ⅞-in. hollow hexagon. These had all done considerable work as dry machines. Three new poppers of lighter weight have also been in use.

The total repair cost for poppers = £185 8s. 6d.

On the total tonnage raised to date, 107,301 tons, the repair cost per ton for popping is 0.42d.

As a result of improvements in the ⅞-in. popping drill steel repair costs for these machines have lately been considerably reduced.

It is not anticipated that the repair cost per ton for popping will increase, for, as mentioned, most of the jackhamers had been in commission prior to this record.

Considering stope machines, it appears that 1d. per ton would cover the repair cost. Taking poppers at 0.5d. per ton, it would seem that for the conditions on the mine 1.5d. per ton would cover the repairs of all machines and poppers used in stoping.

Probable Life of Hammer Drills.

The maximum tonnage broken to date with a machine using 1¼-in. hollow round steel is 9052.

The total tonnage a machine of this class will break under conditions here, before being discarded, might conservatively be taken at 20,000 tons. With a first cost of £93 per machine, the cost per ton to cover this would be 1.11d.

If the life were 25,000 tons, the cost per ton would be 0.88d., or, at 30,000 tons, 0.77d.

The maximum tonnage broken to date with a machine using 1-in. hollow hexagon steel is 6552. If the life is 12,000 tons, the cost per ton to cover first cost will be 1.3d. If the life is 15,000 tons, the cost per ton to cover first cost will be 1.04d. ; and if the life is 20,000 tons the cost per ton to cover first cost will be 0.78d.

It would seem, therefore, that for stoping machines, excluding popping, the total cost per ton would approximate—

First cost	1d.
Repair cost	1d.
	—
Total	2d. per ton.

The cost per ton to provide for replacement of poppers can only be estimated—say 0.5d. per ton, or 1d. per ton to cover first cost and repairs of poppers. The total then for all machines and poppers for first cost and repairs would be 3d. per ton.

Reciprocating Drill Repairs Prior to May, 1919.

The following figures were available for the repair costs of reciprocating drills prior and up to April, 1919. Many of the spares for these were made at the mine workshop.

From April, 1917, to April, 1919, inclusive, the cost of repairs in the shop was £1226 13s. In this period, owing to strikes, there were a little under 23 working months. Taking 23 months, the monthly shop repair cost would be £53 6s. 8d.

During 1918 the average monthly consumption of new spares (purchased mostly in 1916 and 1917, from the makers) was £17 12s. 11d. Taking this as normal, the total monthly repairs for reciprocating drills was £70 19s. 7d. (*i.e.*, £53 6s. 8d. plus £17 12s. 11d.

From April, 1917, to April, 1919, inclusive, 277,290 tons of ore were broken, or 12,056 per month for 23 months. Driving with reciprocating drills amounted to 2116 ft. ; cross-cutting to 590 ft. ; and shaft-sinking to 33 ft. Allotting one-third of monthly repairs to development and the remainder to stoping -

Development repair cost per month .. .. £23 13s. 2d.
Stoping repair cost per month .. .. £47 6s. 5d.

Cost per ft. of development (2739 ft. for 23 months, or 119 per month, = 3s. 10.7d. per ft.

Stoping 12,056 tons per month = 0.49d. per ton.

The above figures may include some spares for rebuilding machines thus a portion of first cost may also be represented, but this is uncertain, and is not taken into account.

Otherwise, taking the average life of a piston drill at three years, on three shifts, and the first cost new at £50, a sum of £1 7s. 9d. per month for 36 months would cover first cost per machine.

With an average number of 11 reciprocating drills stoping, £15 5s. 3d. per month would be required to cover their first cost. For 12,056 tons per month, this is equivalent to 0.30d. per ton.

Taking 0.49d. per ton as purely repairs, the total per ton first cost and repairs is 0.79d. (*i.e.*, 0.49d. plus .30d.) These costs would be higher if the increased prices of machines and spares, above those ruling in the earlier part of the above period, were taken into account. The total cost per ton, first cost and repairs, would then probably be not less than one penny per ton.

No attempt has been made to estimate the total development cost with these machines.

It will be seen, therefore, that if the foregoing estimates are reasonably correct, on this mine the total cost per ton (first cost and repairs), of hammer drills for stoping, is 2d. per ton, as compared with 1d. per ton for reciprocating drills. Including popping, which would be much the same in each case (rather higher with internal water feed), at 0.5d. repair cost, and an estimate of 0.5d. per ton for first cost, the total machine costs would be respectively 3d. per ton now and 2d. per ton previously.

The increased boring speed, lighter weight, reduced air consumption (less per minute, but chiefly reflected in the fact that the air-consuming time per ft. of boring is less) on the above figures show a clear credit to the hammer drill.

HAMMER DRILLS USING 1¼-IN. HOLLOW ROUND STEEL COMPARED WITH 60-LB. JACKHAMERS MOUNTED FOR STOPING.

Indications point to the fact that the heavier machine, using 1¼-in. hollow round steel, will be the more general stoping machine. There is, however, a field for the mounted jackhamers for this work.

In conclusion, it may be stated that in these notes much detail has been given which may be wearisome, but it is a matter of some difficulty to attempt a comparison of the work of various machines without giving a fairly full account.

Proceedings Aus. I.M.M. (Inc.)] [N.S., No. 52, 1923.

THE HYDROGEN ROASTING OF PYRITIC ORES WITH RECOVERY OF SULPHUR AND IRON.

By Hartwell Conder, M.A. (Camb.), A.R.S.M. (London).

In the ordinary course of treatment of pyritic ores or concentrates, whether for the recovery of copper, tin, gold, or any other contents, the removal of some or all of the sulphur contents is almost invariably the first step. The practice is to roast the ore in a current of air so that the sulphur is oxidized to sulphur dioxide and the metal constituents are changed from sulphides to oxides. If it is feasible to produce sulphuric acid in conjunction with the treatment process, the sulphur can be usefully employed; but if this is not possible, then the sulphur contents not only become a dead loss, but tend to be a nuisance as well in the surrounding neighbourhood.

The problem of avoiding this disadvantage and making economic use of the sulphur has had attention from the early days of scientific metallurgy, and in the past few years very special efforts have been made both to drive part of the sulphur as such from pyrites and recover it, and to replace the sulphur in the SO_2 by carbon, with the production of carbon dioxide and precipitation of the sulphur. So far economic success does not appear to have crowned these efforts.

In the case of pyrites, of the two atoms of sulphur which go to make the FeS_2, the first is very lightly held. Dissociation is effected by heat alone at a temperature which is given by Peters as 516° C. Now, sulphur and hydrogen will unite or burn together, with the production of sulphuretted hydrogen. It occurred to the writer that, if hydrogen was passed over pyrites heated to the above temperature, this first atom would be snatched away by the hydrogen, and could possibly be economically recovered.

The first step consisted of some simple laboratory experiments. From these it was found that this first atom was readily removed, and that the extraction was practically complete. Further, the reaction took place at a low temperature, a dull red heat being sufficient.

The next step was to inquire whether there was any economic hope of extracting the sulphur profitably. During the past few years the production of hydrogen on a large scale has been widely developed. Of the several processes, the electrolysis of water stands forth as the cheapest where power is available at a low cost, and a safe figure to take for the relationship between power and gas is 6 cub. ft. per kw. hour. Now, in certain places in Australia water power can be secured at a cost of under $\frac{1}{10}$d. per kw. hour, and this figure has been taken as the basis of the cost of the hydrogen in the process. If this is carried out to the sulphur it means a cost per ton of sulphur of £1 14s. 8d. The parity value of sulphur on the West Coast of Tasmania has been of late about £5, so that there was a margin for other costs than hydrogen of £3 5s. 4d. This margin was not large, but it was enough to give encouragement for further work, and a small experimental plant was installed. The first was a vertical tube through which the powdered ore dropped to a chamber where it encountered an oxyhydrogen flame with excess of hydrogen. This was not a success. In spite of lagging, the escape of heat was great, and the hydrogen was diluted with water vapour, so that the grade of exit gas was low. The idea throughout was to rely on the very cheap power for everything—heat, gas, and movement; but in small plants the loss of heat is disproportionate, and the drain on the gas supply was too great.

A small charcoal furnace was then installed, and a wrought-iron tube of 4-in. diam. was heated in this. The tube was inclined, and a rabble, moving backwards and forwards through 180°, advanced the ore from the top to a delivery tube at the bottom. With this rough appliance the grade of exit gas was brought to 90 % H_2S; the extraction was brought close to theoretical; the character of the roasted ore was found to be satisfactory; and the corrosion of the ironwork was slight. On the other hand, it was very difficult to control the temperature, and if this ran up there was fusion of FeS, which blocked the work. Also, a small amount of sulphur was driven off as such, which condensed in the upper part of the retort and tended to consolidate the feed.

One point was made clear by this work, and that was that a certain proportion of hydrogen would pass on with the H_2S. This had an important bearing on the work. There is a process working in

England called the Claus Chance process. Sulphur existing as CaS occurs as a waste product from the alkali works. This sulphur is converted to H_2S and then passed over heated ferric oxide in a Claus Chance furnace with a certain limited proportion of air. The H_2S burns to H_2O and sulphur, and this latter is recovered. The process is automatic and cheap, and the writer, in the early stages, had relied on it to recover his sulphur from the H_2S ; but the presence of the hydrogen in considerable proportion blocked this. The mixture would be too explosive, while the removal of the hydrogen would involve too much loss and expense.

Incomplete, therefore, as the retort experiments were, they showed that it was essential to look for other means of recovering the sulphur. There were several possible lines of inquiry :—

(1) One that seemed to promise well was the combination of sulphuretted hydrogen, from the hydrogen roast with sulphur dioxide produced by continuing the roast with air, or the oxygen produced electrolytically with the hydrogen.

$$2\ H_2S + SO_2 = 2\ H_2O + 3\ S.$$

This method is given in Lunge's volumes on the alkali industry, and a good many patents have been taken out for modifications of it. The main trouble seems to have been the uniting of the water formed with the gases to produce every variety of sulphur acids. The sulphur, too, precipitates in a form difficult to collect. The favourable factor is that another 25 % of the sulphur is theoretically recoverable, or 75 % in all. On the whole, though, this method seemed to carry too many doubtful points, and was passed out.

(2) A very pretty method was to pass the H_2S into a solution of iodine in potassium iodide. The iodine displaces the sulphur with the formation of hydriodic acid and precipitation of sulphur. By electrolysing the solution both hydrogen and iodine are regenerated, and the decomposition voltage of hydriodic acid is only .5, so that the power consumption would be greatly reduced. In spite of the high cost of iodine, this process seemed promising, but the sulphur was deposited in a plastic, rubber-like form, carrying much of the solution with it. The only means of separation seemed to be heat, and that was fairly certain to produce iodides of sulphur and entail loss of iodine.

(3) Chlorine and sulphuretted hydrogen unite to produce sulphur and hydrochloric acid. Since the electrolysis of sodium chloride or

of hydrochloric acid would yield both hydrogen and chlorine, this method was feasible, but the formation of sulphur dichloride was anticipated, and occurred to such extent that this line of action was abandoned.

(4) Text-books state that sulphuretted hydrogen is dissociated by heat, and the temperature of dissociation is given as 400° C. On the strength of this some work was done with heated tubes, and a line of attack with incandescent platinum gauze was plotted out. At the same time search was made of the authority for the original statement, and it was pushed home to some rather crude work by Myers. Further search revealed more recent work by Preuner in 1907, based on the diffusion of hydrogen through heated platinum, and the results showed that, while dissociation started at about 400° C., it required very high temperatures to secure even a 50 % dissociation. The preliminary tube work seemed to confirm this, and the method appears to be impracticable.

(5) Text-books, again, state that H_2S is dissociated by the electric spark. Elaborate search revealed no details of any work on this gas, but some interesting data were found of the dissociation of water vapour by high tension sparks, which gave a basis to start on. A large coil was employed which could give a spark over a gap of 13 in. The spark was reduced in length, and platinum electrodes were introduced into a glass tube of wide diameter, through which the H_2S was passed. Sulphur was readily precipitated and condensed on the glass. The amount of current and the weight of sulphur were recorded. At first the amount of current was far beyond economic limits, but with changes of voltage and conditions the results were brought to a standard which gave considerable promise. This method may yet prove to be useful as a preliminary treatment for the high-grade gases before passing them on to more complete extraction.

(6) The last method consisted of passing the H_2S into a solution of ferric chloride :—

$$H_2S + 2\ FeCl_3 = 2\ FeCl_2 + 2\ HCl + S.$$

This was tried years ago in gasworks practice, and an attempt was made to regenerate the $FeCl_3$ by passing air through it. In the present day the regeneration can be effected readily by electrolysis of the solution. The decomposition voltage of HCl is low, being

1.3, so that power costs are kept down and the hydrogen is regenerated for the process. By keeping the solutions at proper strength the sulphur is precipitated in granular form, which settles and washes easily.

The outcome, then, of the work was that hydrogen was produced by the electrolysis of a solution carrying hydrochloric acid and ferrous chloride, yielding ferric chloride. The hydrogen is passed over the pyritic ores or concentrates, yielding sulphuretted hydrogen. This gas, with or without preliminary electrical treatment, is passed into the ferric chloride solution, yielding ferrous chloride and hydrochloric acid, and the process becomes cyclic.

The success of a process is measured in economic terms. As shown above, taking the cost of power as .1d. per kw. hour, the supply of hydrogen involves an expense of £1 14s. 8d. per ton of sulphur produced, or 4.16d. per unit of sulphur. With sulphur at £5, each unit is worth 1/.

In trying out the costs, one is warranted in making the conditions as favourable as possible, and the material kept in view is either flotation concentrates, as at Lyell, or pyritic tin concentrates, as found on the West Coast. These are already in the condition for treatment, and carry 15 % to 20 % loose sulphur, the removal of which entails expense now.

The process involves the following steps :—

(*a*) Supply of hydrogen.
(*b*) Roasting of ore.
(*c*) Precipitation of sulphur from H_2S.
(*d*) Regeneration of solution.
(*e*) Conversion of precipitate into market form.

Of these, the cost of the supply of hydrogen is known approximately, and carries the regeneration of the solution. The precipitation of the sulphur means only pumping costs, which at low power rate would be very small. Conversion into market form appears only to involve simple handling. There remains the roasting of the ore, and in the writer's opinion the success of the process hinges absolutely on this. Unless this phase can be carried out automatically and continuously, the total costs will be too high. If this can be achieved there is a margin.

The favourable features are the low temperature, the ready reaction, the smooth flow of the product, freedom from dusting (owing probably to the low specific gravity of the hydrogen), the low specific heat (.13, Liddell) of pyrite, the low heat of dissociation of FeS.S (which probably is less than the 4800 units of H_2–S. The adverse factors are that a closed retort must be used; the hydrogen gas is explosive if air gains access to it; the sulphur fumes tend to corrode any ironwork; sulphur may be driven off as such and clog the feed and the outlet and any moving parts.

The design of a furnace to meet these conditions was the next step in the research, but by this time the writer had neglected his ordinary business for as long as was feasible, and it was necessary to call a halt. Application for assistance to the Department of Science and Industry met with every courtesy, but the trouble was "No funds," and so the matter rests to-day. All that is possible is to outline the form of furnace that seems most feasible, and invite criticism and suggestions from the members of Aus. I.M.M. before it is installed.

Working on the principle that the maximum use must be made of the cheap power available, some form of electric furnace seemed inevitable. Coal would be almost as costly per unit of heat contained as electricity; wood might be less, but in each case the heat would have to be applied externally, and loss, both in burning and in traversing the casing of the retort, would be prohibitive. It was decided to endeavour to make a retort which could be heated internally by electrical resistance. Since this idea was adopted the writer has found that retorts 6 ft. long and 12 in. diam. are being heated thus by nichrome strips. In this case the sulphur fumes would probably make the cost of nichrome too high, and it was contemplated to use cast-iron; its resistance is 50 times that of copper, and renewals would be cheap. It is premature to attempt to give all the details, but the furnace itself would be moulded out of a mixture of fireclay, cement, and waterglass, and the walls would be made thick to enclose the heat. This mixture has been found to stand up to the temperature and fumes quite easily. The ore would be rabbled through the furnace and the central axis would be water-cooled. The rabbles themselves would be of the same cement mixture, with iron central boss and reinforcement of iron. The wear should not be great, and repair would be simple.

The feed would be automatic. It would be necessary to dry the material first in a drier on the same design as the retort; this is not included in the estimate of costs, because in any process of treatment the heat for this drying must be provided for. The delivery would also be automatic. It is hoped that the whole action will proceed continuously, and that the wear and tear will be low.

The actual heat units required to raise 1lb. of ore from 50° to 961° F. are 118.4 B.Th.U. In theory 1 unit of electricity gives 3443 B.Th.U., so that for a ton of ore 77 electrical units are required. With 83 % efficiency and .1d. cost, this gives a cost of about 9¼d. per ton for heat.

Wear and tear is taken at 1s. per ton of ore.

If a plant be considered to treat 50 tons of ore per day, of which 20 % is recoverable sulphur, the following is an outline of the probable cost. For the supply of hydrogen 2000 electrical horse-power is required; for heat, 250; sundry, 50, or a total of 2300.

Power—2300 electrical h.p. at .1d. per unit	£23	0	0
Wear and tear—1s. per ton	2	10	0
Interest on cost of plant apart from power plant—8 % on £50,000	10	19	2
Balance—for labour, etc.	13	8	0
	£49	17	2

The value of 10 tons of sulphur at £5 is £50. The whole crux is the labour. If this can be cut down to a man and a boy per shift, with a spare man on the day shift, the wages bill will be about £6, leaving a balance of £7. This should be more than sufficient to cover the sundry expenses of marketing the product and losses of gas in the cycle and to leave a margin to add to the 8 %.

No charge has been allowed for supervision, because it is realized that the scope of the process belongs, as an adjunct, to the main lines of treatment of ores. Its province is to rid flotation concentrates of their excess sulphur and to remove sulphur from tin ores where such work is now being done with complete loss of all the sulphur content. With any rise in the price of sulphur—which the main producers are aiming at—the scope could be greatly extended.

The margin is admittedly small, but a fair comparison is with a cyanide plant as an adjunct to a gold battery. Such plants have in many cases lengthened the life and enhanced the profits of a mine when the small margin of profits could only be wrung from the ore by the most careful and economical handling. It would seem worth while in this case to push the matter forward another step, at small expense, to prove whether like results may not be attained in the recovery of sulphur.

Recovery of Iron.—The second part of the process deals with the recovery of the iron. Pyrites itself is insoluble in hydrochloric acid. Iron sulphide, or FeS, readily dissolves, yielding sulphuretted hydrogen gas ; it is the ordinary method of producing the gas in the laboratory.

Now, the first part of this process aims at recovering the first atom of sulphur, and in doing so it converts the pyrites into plain iron sulphide. If a solution of hydrochloric acid and ferrous chloride be applied to this iron sulphide we have the following reaction:—

$$2\ FeS + HCl = FeCl_2 + H_2S.$$

The hydrochloric acid is neutralized, and the solution consists of ferrous chloride alone. If this be electrolysed, iron is deposited, and the solution becomes one of ferric chloride. Now, the first reaction yielded H_2S as well ; if that H_2S be passed into this ferric chloride solution, then, as was shown in dealing with the sulphur alone, we have sulphur deposited and the ferric chloride reduced to hydrochloric acid, and ferrous chloride ready to dissolve more ferrous sulphide.

The process is on similar lines to those followed by Mr. Eustis in America. He, however, treats the ore itself with ferric chloride, and the sulphur is deposited with the gangue. In the process outlined above there is no ferric chloride in the dissolving solution, but free hydrochloric acid. Which method would prove superior in practice only careful comparison could reveal. For either process the preliminary hydrogen roast is especially adapted, since pyrites is impervious to ferric chloride, even as to hydrochloric acid, though some forms of pyrrhotite are amenable in the crude to both.

It has not, so far, been possible to follow up this process beyond the experimental stage in the laboratory. From the work done there no special difficulties appeared to stand in the way. The process would resemble, in its main features, that of the Electrolytic Zinc Co., at Hobart, and from the chemical and electrical aspect should be simpler. The value of the product is much less, but the cost of the ore would be proportionately lower, and in some cases might be available as a waste product.

The whole question of the economic side of the problem was given a preliminary consideration by Mr. D. Avery for the author and his friends, and the conclusion arrived at was that a plant to treat 100 tons per day would cost in the vicinity of a million, and that with iron at £7 per ton and sulphur at £5, and a small percentage of tin, a working profit of £263 per day might be looked for, out of which amortisation and interest on capital must be provided.

The vital point is the value of the iron produced. Pig iron in England sells now at about £5 per ton, which is equivalent to about £8 per ton c.i.f. &c. landed in Australian ports. Australian pig iron if about the same price delivered in the main Australian ports. Steel ingots imported into Australia, 1921–22, were valued at about £13 per ton; steel bars and sheets are quoted round about £20 per ton now. The question is, What valuation can be placed on electrolytic iron as it comes in sheets from the vats? and so far no definite figure has been arrived at. In its favour may be placed the fact that in France works are stated to be running to produce electrolytic iron from impure iron, since special uses have been found for it. Also, Swedish iron and Lowmoor iron command prices more than 100 % above the ordinary. Again, too, it has been predicated throughout that power must be at a minimum to make this process feasible. If such power is available it will be a most important factor in any process of manufacture from the first product to the marketable article. In Melbourne steel is now produced from the electric furnace and manufactured into plates and rods with power at a cost far above what is here contemplated. With this advantage in the secondary processes, an enhanced cost in the primary can be faced and overcome.

A design was roughed out for a small plant to treat 100 lb. of ore in 24 hours. The power required would be about 10 kw. hours, and the cost about £400 to £500. A great deal of detail was gone into with regard to the form of the electrolytic cells and the character of the precipitating towers. Towers 15 ft. high, with splash boards from top to bottom, were finally decided upon. Barrel precipitation has been used in similar work in America, but does not appear necessary here. In the electrolytic cells diaphragms will be required for those depositing iron, and canvas diaphragms were used in some work done at Yerington, Nevada, and proved satisfactory. It is not clear whether diaphragms will be required in electrolysing the HCl, $FeCl_2$ solution; if they can be avoided it will greatly reduce the power consumption. All such work, however, would be regulated and modified as experience was gained with the experimental plant. What the author feels is that the price of coal and the cost of labour is likely to increase rather than fall back again in the next 20 years. It is practically certain to increase in the next 200. Sooner or later a process such as this, which is economical in both power and labour, and whose main burden is the capital outlay, will assert itself against its present competitors and ultimately will pass them in the race. If this work helps to bring that day a little nearer it will not have been done in vain.

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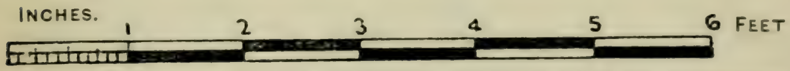

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